EXAMINATION OF GEOLOGICAL INFLUENCE ON MACHINE EXCAVATION OF HIGHLY STRESSED TUNNELS IN MASSIVE HARD ROCK

by

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ABSTRACT

A combined geological and rock mechanics approach to tunnel face behaviour prediction, based on improved understanding of brittle fracture processes during TBM excavation, was developed to complement empirical design and performance prediction for TBM tunnelling in hard rock geological conditions. A major challenge of this research was combining geological and engineering terminology, methods, and objectives to construct a unified Geomechanical Characterisation Scheme. The goal of this system is to describe the spalling sensitivity of hard, massive, highly stressed crystalline rock, often deformed by tectonic processes. Geological, lab strength testing and TBM machine data were used to quantify the impact of interrelated geological factors, such as mineralogy, grain size, fabric and the heterogeneity of all these factors at micro and macro scale, on spalling sensitivity and to combine these factors within a TBM advance framework. This was achieved by incorporating aspects of geology, tectonics, mineralogy, material strength theory, fracture process theory and induced stresses.

Three main approaches were used to verify and calibrate the Geomechanical Characterisation Scheme: geological and TBM data collection from tunnels in massive, highly-stressed rock, interpretation of published mineral-specific investigations of rock yielding processes, and numerical modelling the rock yielding processes in simulated strength tests and the TBM cutting process. The TBM performance investigation was used to identify the mechanism behind the chipping processes and quantify adverse conditions for chipping, including tough rock conditions and stress induced face instability. The literature review was used to identify the critical geological parameters for rock yielding processes and obtain strength and stiffness values for mineral-specific constitutive models. A texture-generating algorithm was developed to create realistic rock analogues and to provide user control over geological characteristics such as mineralogy, grain size and fabric.

This methodology was applied to investigate the TBM chipping process to calibrate the Geomechanical Characterisation Scheme. A Chipping Resistance Factor was developed to combine the quantified geological characteristic factors and laboratory strength values to predict conditions with high risk of poor chipping performance arising from tough rock. A Stress-Related Chip Potential Factor was developed to estimate conditions with high risk of advance rate reduction arising from stress-induced face instability.
COAUTHORSHIP

The following thesis represents the original work of the author. Credit, however, is due to Dr. Mark S. Diederichs and Dr. Peter K. Kaiser who contributed both scientifically and editorially.
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be in the way. I took most of their advice and by blending a desire to enjoy what I do and find something useful to enjoy doing, I have finally made it to this point. Their eternal support, which came in so many ways, has allowed me to accomplish my dreams.

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STATEMENT OF ORIGINALITY

I hereby certify that all of the work described within this thesis is the original work of the author. No part of this thesis has been submitted elsewhere for any other degree or qualification. Any published (or unpublished) ideas and/or techniques from the work of others are fully acknowledged in accordance with the standard referencing practices.
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LIST OF SYMBOLS

BTS  Braziliang tensile strength
B_x Britleness index (x = index number)
c Cohesion
CAG Central Aar granite
CR Chipping resistance
DI Drillability index = penetration rate/thrust
\( \varepsilon \) Strain
\( \varepsilon_e \) Elastic strain
\( \varepsilon_p \) Plastic strain
\( \phi \) Friction angle
F_A Fabric factor
FAD Microlithon spacing factor
F_G Grain size and grain size distribution factor
F_GD Grain size distribution factor
F_GP Petrographic grain size
F_GT Tectonic grain size (i.e. subgrain boundaries, etc.)
FLAC Finite difference continuum numerical modeling code (Itasca)
F_M Mineralogy factor
F_MA Accessory mineral factor
F_MM Major mineral factor
F_SS Spalling sensitivity
F_SSA Stress-related spalling sensitivity
\( i_o \) Basic penetration rate
\( \mu \) Coefficient of friction (tan\( \phi \))
M_1 Critical thrust
MPO Mineral preferred orientation
NAR Net advance rate = advance rate (per minute) during active excavation
PLT Point load index strength
ROC Receiver/Response Operating Characteristic (Curve)
\( \sigma_1 \) Intermediate principal stress
\( \sigma_1 \) Major principal stress
<table>
<thead>
<tr>
<th>Symbol</th>
<th>Description</th>
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<tr>
<td>$\sigma_3$</td>
<td>Minor principal stress</td>
</tr>
<tr>
<td>$\sigma_c$</td>
<td>Compressive stress; unconfined compressive strength</td>
</tr>
<tr>
<td>$\sigma_t$</td>
<td>Tensile stress</td>
</tr>
<tr>
<td>SAG</td>
<td>Southern Aar granite</td>
</tr>
<tr>
<td>$S_{CP}$</td>
<td>Stress-related factor</td>
</tr>
<tr>
<td>TBM</td>
<td>Tunnel Boring Machine</td>
</tr>
<tr>
<td>UCS</td>
<td>Unconfined compressive strength</td>
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<tr>
<td>$\tau$</td>
<td>Shear stress</td>
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For my parents
Chapter 1: Introduction and Executive Summary

1.1 In-situ Rock Behaviour

Several methods exist for designing underground rock excavations and the tools and approaches required to undertake the excavation process. All of these are aimed at predicting in-situ rock behaviour and for the most part are based on empirical relationships between actual behaviour and features of both the excavation and the geology (i.e. Bruland, 1998), or on semi-empirical relationships between rock mechanics parameters measured by laboratory testing and models, usually verified by case histories (i.e. Barton, 2000; Rostami and Ozdemir, 1993).

These methods have proved to be adequate in most situations but there remains a need and desire to understand the rock failure process to be better able to predict rock behaviour. One logical avenue would be to employ further laboratory rock strength testing for innumerable situations and observe how a rock’s underground behaviour compares to the strength test results. Another approach, which was rigorously applied in the 1960s and 70s (i.e. Brace, 1960; Brace, 1961; Illston, Dinwoodie and Smith, 1979; Lama and Vutukuri, 1978; Tapponier and Brace, 1976), is to return to fundamental materials science principles to understand the failure process of polycrystalline materials with a variety of physical and mechanical properties.

By understanding the fundamentals of how rocks fail, it will be possible to understand the failure process during laboratory testing as well as during excavation and to better relate laboratory test results to in-situ behaviour. The research presented in this thesis contains a study of the impact of a set of geological characteristics on in-situ rock behaviour, and its comparison to lab strength test values. The aim is to provide a method by which lab strength values can be translated into better in-situ rock behaviour predictions with respect to TBM performance.

Using the information uncovered by fundamental rock fracture processes a classification scheme could be developed for rock behaviour prediction. Miall (1999) makes a case for using classification schemes in the geological sciences, saying that: “…they attempt to create order out of apparent chaos…” and that the “… best classification schemes are those that highlight differences in origins of geological features, are therefore particularly useful for purposes of prediction and extrapolation…”. He wrote about sedimentary rocks, but this is equally applicable to classification for engineering purposes, especially if the classes are based on impact on particular rock mechanics processes.
In order to work out the fundamentals for the classification scheme and test them, data were obtained from various sources: published relationships between rock mechanics processes and geological characteristics, TBM excavation data and numerical modelling results. Disparate published relationships between laboratory strength, fracture processes, such as initiation and propagation, and numerical modelling, and various geological features, such as mineralogy, grain size and fabric were examined for evidence supporting the selection of key geological characteristics for rock yielding processes. For the TBM excavation data a new set of data analysis methodologies were developed to identify different rock mechanics processes and rock behaviours encountered during excavation. For the numerical modelling, a new methodology was developed to simulate realistic rock analogues in which geological parameters could be varied to represent rock characteristics and heterogeneity at the grain scale. All of these investigations culminated in the development of a Geomechanical Characterisation Scheme used to predict rock behaviour with respect to brittle fracture and in particular for chipping performance prediction for TBM excavation and tunnel face stability.

1.2 Research Focus

The factors affecting in-situ rock behaviour are numerous and in order to conduct a thorough investigation, the possible range of geological features studied must be narrowed. The focus of the research is the failure behaviour of intact rock, in particular the creation of new fractures through intact rock as a result of in-situ stress induced by the excavation or excavating tools. The rocks investigated in this study are massive, crystalline rocks with similar mineralogy. The use of the label ‘massive’ for the rocks used in this study refers to rocks with little to moderate jointing, for example with GSI > 70 (Figure 1.1) and RMR > 75, and range of in-situ stress from low to high (Figure 1.2). Other researchers have spent considerable effort understanding the behaviour of fractured rock masses where the characteristics of discontinuities dominate the behaviour (i.e. Barton, Lien and Lunde, 1974; Bieniawski, 1989; Hoek, 1994), and these rock mass types are not addressed in the present research.

The composition of crystalline and clastic rocks differs mainly by the presence of a cementing matrix that holds the individual clasts together in clastic rocks, as opposed to the growth of individual minerals, which eventually interlock to some degree in crystalline rocks. During the geological history of a rock, tectonic deformation may have decreased the degree of interlocking but as long as cohesion has been maintained they are treated as crystalline rocks in this study. The most common minerals are used as they are representative of the majority of
rocks that are encountered during excavation, and reflects the available rock samples used in this study. Some extension of the mineralogy is possible, but highly altered rocks, those that have been deformed to the point of no longer having cohesion, and jointed rock masses are better addressed by other methods.

![ROCK MASS CHARACTERISTICS FOR STRENGTH ESTIMATES](image)

Figure 1.1: GSI range of this study (Marinos et al., 2005)

### 1.3 Terminology

**Brittle Yield behaviour** is a term used in this thesis to describe the way in which fractures are initiated and propagate through a rock in response to applied stress. It is highly dependent on several factors including stress conditions and geometry, but the main focus of the research is to identify the geological factors that impact fracture behaviour in an underground environment.

**Yield strength** is the maximum stress the rock is capable of withstanding.
Yield strength at excavation boundaries is used throughout this thesis to describe the strength experienced under the geometrical conditions of underground excavations, in contrast to the strength determined by lab yield strength testing, which is highly dependent on the geometry of the test.

Introduced in Chapter 2, a Tunnel Boring Machine (TBM) is a form of mechanical rock excavator that performs excavation by a circular, rotating head on which circular cutters are installed perpendicular to the head and are the only portion of the rotating head that normally come in contact with the rock. The cutters are installed on bearings and roll in response to the rotation of the head. The excavation of the rock is normally achieved by the pressure induced by the cutters and the generation of fractures through intact rock.

Spall sensitivity and Chipping Resistance are introduced in Chapter 4. They are used to describe the evaluation of the effect of the combination of different geological factors on the in-situ behaviour of the rock. Spall sensitivity describes the isotropic susceptibility of rock to failure by spalling, which is a spontaneous failure of intact rock as a result of applied stress. The chipping resistance combines the spall sensitivity and any anisotropy and the orientation of applied stresses with laboratory strength values to determine the likelihood of fractures being generated through the intact rock.
A common term used to describe the strength characteristics of a rock with respect to TBM excavation is its *chipping performance*, that is, the ease by which a TBM cutter can penetrate the rock and generate chips. One major aspect of this research is describing the geological features that contribute the most to improving or impeding a rock’s susceptibility to cutting.

*Stress induced face instability* is used to describe the generation of new fractures through intact rock at the tunnel face in response to the induced stress condition. It is to be differentiated from blocky face conditions resulting from falling blocks created by the intersection of pre-existing discontinuities. Both conditions may result in blocks falling off the face into the space between the TBM head and the tunnel face, the implications of which are discussed in Chapter 3. A second major aspect of this research is delineating the geological conditions under which stress induced face instability is possible.

### 1.4 Approach

The approach to undertaking this work can best be summarised as: identification of a hypothetical solution, validation with a real dataset, calibration with synthetic data, and verification of the solution with the real dataset. A combination of fundamental and empirical approaches were used to determine the important geological factors for estimating in-situ strength; these were validated for a limited real dataset containing geological and TBM machine performance data. They were then calibrated using numerical modelling of the fracturing process with explicit geological factors as the inputs (see Appendix 1.1), and were then verified using the limited real dataset and three other sparse datasets containing geological and TBM machine data. The use of synthetic data for calibration is especially useful to investigate a wide range of properties, in this case geological characteristics, for which real data are only available for a limited range of properties; in this case a large dataset representing similar rock types with narrowly varying geological characteristics (Cundall, 2002).

A thorough investigation into the fundamental research available concerning rock fracturing and the parameters that have the greatest impact on the fracture behaviour and strength of rocks was performed. The conclusions from this investigation were used to identify three factors that describe the geological nature of a rock, that can be interpreted by sample analysis combined with literature review, and that have the greatest effect on rock fracture, which were combined into a Geomechanical Characterisation Scheme.
An empirical analysis of the geological characterisation of rock samples collected in a Swiss tunnel excavated by TBM compared to the performance indicators of the TBM was used to test the sensitivity of the TBM performance to each parameter and validate their selection for the Geomechanical Characterisation Scheme. Numerical modelling of simple strength test models (two dimensional Brazilian and UCS test models) provided a test module for investigating the precise influence of each parameter on rock fracture where multiple tests can be conducted under conditions that can be controlled by the user.

Further numerical modelling of the TBM cutting process and stress induced fracture generation at the tunnel face provided a secondary test module in which the impact of spall sensitivity and fracture potential on TBM performance were investigated and used to calibrate the Geomechanical Characterisation Scheme. This analysis provided information that could be verified by available data, as well as to extrapolate to different conditions that were not encountered during data collection. A final verification of the Geomechanical Characterisation Scheme was performed using rock samples and TBM performance indicators collected in three additional Swiss tunnels excavated by TBM.

1.5 Objectives

The objectives of this research are two-fold: improvement of geological characterisation to better predict fracture behaviour and in-situ strength of rock, and the successful application of a geological characterisation system to TBM performance in massive moderately to highly stressed rock.

1.5.1 Fracture Potential

The main objective of this portion of the research is to identify and quantify the geological parameters that dominantly control the rock fracture behaviour and in-situ strength. While considerable work has been completed on development of indices for rock strength based on laboratory testing combined with empirical and theoretical studies (i.e. Hoek, 1990), a method by which a more precise determination of fracture behaviour and rock strength depends on understanding the impact of the geological characteristics of the rock.

A wide range of lab test values can be narrowed or broken into smaller, more specific groups or units by careful geological characterisation of the parameters that define their similarities or differences. By developing the theoretical basis and functional application of a
methodology by which this can be achieved, a better understanding of fracture behaviour and in-situ strength is possible at the academic and industrial levels.

### 1.5.2 TBM Performance in Deep, Massive Rock

Tunnelling in deep, massive rock provided information concerning TBM performance during excavation of intact rock that may or may not be subject to induced stress damage at the face. The locations of the tunnels in the Swiss Alps (Figure 1.3) provided a geological environment in which tectonic history is a large component of the characteristics of the rocks being studied, which have a wide range of textural features, but a narrow range of mineralogy.

A demonstration of the applicability of a new characterisation method to engineering is important for legitimising the motivation for the work and the results of the work itself. The realm in which this research was conducted determined what aspects of rock mechanics would be investigated, the material that would be used for development and the application of the results. The determination of the impact of the fracture potential on TBM performance in terms of cutting intact rock and sensitivity to stress induced fracturing in the tunnel face is the main objective in this portion of the thesis.

Cutter wear is also interconnected to the chipping process by both being affected by the chipping process and affecting the chipping process. Cutter wear is critical to TBM utilisation and performance, especially in very hard, massive ground. Investigations into the geological and TBM operational factors, among others, that affect cutter wear, and how cutter wear affects the chipping process is not within the scope of this research, although it must be noted that it is critical to TBM performance.

Figure 1.3: Location map of the Lötschberg and Gotthard tunnels overlaid on a simplified geological map of tectonic units for Switzerland.
1.6 Synopsis of Findings

The findings of this study are separated into the geomechanical characterisation, which is independent of TBM excavation and could be used in other rock mechanics applications, and the results that are directly applicable to TBM performance prediction and design.

1.6.1 Geomechanical Characterisation

The literature review has shown that mineralogy, grain size and grain size distribution, and fabric intensity and orientation are all important factors influencing rock strength and fracture behaviour. Several studies have been undertaken to quantify the magnitude of the influence, especially on laboratory test strength, but few have addressed more than one or two factors simultaneously. The survey of previous work did show, however, that among the multitude of geological characteristics possible, the three selected for this research dominate spalling sensitivity ($F_{SS}$ in Figure 1.4, see definitions) and chipping resistance ($C_R$ in Figure 1.4, see definitions).

The review of previous work concerning the fracture process, especially in granite, shows that different minerals impact the strength of the rock differently. The mineralogy was combined in a format in which the relative percentages of strong, stiff, dominant minerals are used to quantify a factor that affects spall sensitivity ($F_M$ in Figure 1.4). The platy, weak, or extremely stiff minor minerals are also combined to modify this factor. TBM performance data analysis and numerical modelling showed that increasing mica content increases the ability to propagate fractures, while very low mica contents (less than 5-6%) will have a similar, although less pronounced, impact.

A reanalysis of published data, combined with conclusions from previous work demonstrates that increasing grain size decreases strength and increases the potential for uncontrolled fracture propagation. A second factor is presented to account for the contribution of the grain size and distribution to isotropic spall sensitivity ($F_G$ in Figure 1.4). This was confirmed with numerical modelling of samples with varying grain size and comparison of grain size characteristics to TBM performance.

The evaluation of previous studies addressing the impact of fabric on fracture behaviour and rock strength results in the development of a factor that describes fabric intensity and its effect on fracture potential ($F_A$ in Figure 1.4). This factor can be used for nearly isotropic rocks containing several fabric planes or combined with tunnel geometry and fabric orientation for anisotropic rocks with a single fabric plane. Numerical modelling and evaluation of TBM
Figure 1.4: Flowchart demonstrating the Geomechanical Characterisation Scheme

performance data quantified the impact of each fabric type and orientation on chipping performance.

The Geomechanical Characterisation Scheme developed in this thesis is made to allow quantification of each of the geological F-Factors and impact from in-situ stress \( (S_{CP}) \) to modify laboratory strength testing data and obtain a quantified estimate of potential for poor chipping during TBM excavation. Testing of rock types from 4 different tunnels showed that the Geomechanical Characterisation Scheme is capable of this, albeit in some cases conservatively.

### 1.6.2 Chipping Performance and Tunnel Face Stability

Previous work has shown that lab strength data can be used for TBM design, but this study provides a tool to improve the in-situ behaviour of the rock for better TBM design and performance prediction. The chipping resistance factor has been used to determine the chipping performance of the rock under the specific conditions present at the cutter tip. It is also used to assess the potential for stress induced fracturing of intact rock at the tunnel face, leading to instability.

The impact of the variations of both of these aspects of rock behaviour, which are shown to be related, on TBM performance are determined by evaluating the fabric type and orientation.
and the in-situ stress. The Stress-Related Stability factor ($S_{CP}$ in Figure 1.4) allows an estimate of the potential for advance rate reduction due to preconditioning and face instability for different fabric orientation and in-situ stress condition. The $S_{CP}$ assesses face stability-specific interpretations for advance rate reduction arising from face instability. The processes that control the generation of new fractures through intact rock in response to applied stress are manifested differently during TBM excavation depending on the geometry of the tunnel and geology, as well as the induced stress condition. Charts combining fabric orientation and in-situ stress condition, and fabric type and microlithon spacing, are used to identify the potential for advance rate reduction.

### 1.7 Practical Implications

When designing a TBM it is important to determine the likely conditions under which the TBM will have to operate. The best design is capable of efficiently operating through the majority of the conditions encountered and with little risk to the machine and personnel. The development of a practical geological characterisation methodology will provide a tool with which a TBM designer can better predict the conditions under which the TBM will have to operate. Of highest importance is to identify potentially difficult excavation conditions arising either from very tough rock to excavate or from advance rate reduction due to induced instability in the face. The Geomechanical Characterisation Scheme aims at identifying potential zones of tough rock through the $C_R$ value (Figure 1.5). The in-situ stress estimation is capable of identifying potential advance rate reduction arising from induced face instability due to excessive preconditioning of the face (Figure 1.5).

When performing a site investigation and preparing tender documents for TBM excavated tunnels, a characterisation of the rock in terms of mineralogy, grain size and grain size distribution, and fabric intensity and orientation will benefit from the use of the Geomechanical Characterisation Scheme. The spall sensitivity and chipping resistance will provide information regarding the in-situ strength and fracture behaviour of the rock, which can be used, in combination with an estimate of induced stress, to predict the potential for poor chipping conditions during excavation.

Changes to TBM design can be made based on the predicted geological conditions, leading to more precise tender bidding, greater adaptability for changing rock conditions and greater safety for both machine. The financial and safety impacts of improved, more adaptable TBM design cannot be overstated. A better understanding of rock behaviour during underground
excavation, with or without a TBM, has universal rock mechanics implications. A methodology for understanding, or even predicting, in-situ rock behaviour may also be useful for excavation layout design and sequencing, support design and selection, and perhaps eventually for drill and blast excavation.

### 1.8 Thesis Structure

This thesis is structured as a manuscript/traditional thesis. As such it contains chapters in which manuscripts are embedded. The information contained in the chapters is the necessary background, be it from existing literature or more detailed description of the work undertaken in the study, which is not included in the manuscripts.

Chapter Two consists primarily of a literature review of excavation by Tunnel Boring Machine and the current methods by which TBMs are designed, their performance predicted and the behaviour of the rock during TBM excavation is anticipated, and the geological setting from which the rock and Tunnel Boring Machine data were collected. A discussion of the advances made by this research and recommendations for changes in geological data collection and used for TBM design and performance prediction concludes this chapter.

Chapter Three contains a discussion of the tunnelling undertaken in the Swiss Alps and the data collection carried out during fieldwork for this research. The TBM and rock strength
data are presented followed by a rigorous analysis of TBM performance over a 400m length involving highly detailed data collection. The analysis leads to identification of conditions, geological and stress-related, under which different cutting and rock instability processes occur. These are used to classify the face stability and chipping performance at tunnel locations within the dataset.

Chapter Four is a description of the development of the Geomechanical Characterisation Scheme of rocks for in-situ behaviour introduced in Chapter Two. A discussion of the fundamentals of rock fracture introduces the work done in this area, which constitutes a large part of the basis of this study. The process of selecting the important factors used in the characterisation is based on published research in the field of rock mechanics and materials science. A semi-parametric analysis of the sensitivity of rock strength and in-situ strength based on published data, some lab strength data from the rocks used in this study and correlated TBM performance data is used to provide weightings to the factors identified for the characterisation in terms of chipping behaviour.

In Chapter Five numerical modelling is used to calibrate the weightings given to the factors by translating them into parameters that describe the behaviour of rock during lab testing and in underground conditions. UCS and Brazilian tensile test models were created and used to parametrically analyse the different F Factors in terms of their impact on spalling sensitivity.

Chapter Six is a demonstration of how the geological characterisation relates to TBM performance. The TBM performance is broken into two aspects: fracturing of intact rock with TBM cutters and fracturing of intact rock by the stresses induced at the tunnel face. Both of these are related to the in-situ strength of the rock and its sensitivity to spalling behaviour induced by stresses under the cutter, at the tunnel face and a combination of both. Numerical modelling of the cutting process and the conditions at the tunnel face are used to investigate the effect of spalling sensitivity on the processes occurring at the contact between the TBM and the rock. These are compared to the data-based interpretations from Chapter Three and parametric analysis from Chapter Five.

A summary and discussion of the main findings of the thesis and a synopsis of the data collection and analysis methodology are presented in Chapter Seven. The limitations and extrapolations of the results of the research are discussed and future work concerning similar investigations in rock types not directly studied in this research and recommendations for further improvement to the methodology laid out in this thesis are described. The appendices comprise data analysis methodologies, numerical modelling methodologies and sensitivity analyses, background information and data relating to the information presented in the main thesis chapters.
Chapter 2: Introduction to Mechanized Tunnelling in Alpine Terrains

2.1 Understanding the TBM Excavation Process in the Alpine Base Tunnels

This chapter consists of three parts: an overview of the state of practice of TBM design, a description of the geology of the Swiss Alps and an introduction to the approach undertaken to improve the state of the art of TBM design.

2.1.1 The Gotthard and Lötschberg Tunnelling Projects

Switzerland’s NEAT (Transalpine Railroad Links) project aims to improve railway transport throughout Switzerland and more effective transport between Switzerland and its neighbours (Figure 2.1.1). Three new base tunnels, the Lötschberg, Gotthard and Ceneri tunnels, will be constructed through the Swiss Alps as part of this project. Investigation and design for these tunnels began in the early 1990’s and construction is expected to be completed in 2015.

As base tunnels, the Gotthard and Lötschberg tunnels are being excavated below the topographic base of the Alps to a maximum overburden height of over 2500m. In addition these tunnels will figure, respectively, as the world’s first and fourth longest transportation tunnels (Lotsberg, 2003). Due to the tight time schedule for construction and the favourable rock types, one third of the Lötschberg and two thirds of the Gotthard tunnel are being excavated by TBM. The magnitude of these projects makes timing critical and accurate TBM performance prediction, used in completion of excavation prediction, is essential to ensure the project stays on budget.

The Lötschberg tunnel is located in the north-south axis connecting Basel-Bern, Switzerland, to Domodossola, Italy, and thence to other parts of northern Italy. The Gotthard tunnel is located 50km to the East of the Lötschberg tunnel and is part of the north-south axis connecting Zurich and Milan.

The Gotthard tunnel comprises a northern (Erstfeld) and southern (Bodio) portal and three intermediate access adits: Amsteg, Sedrun and Faido (Figure 2.1.2). Each access is assigned a lot from North to South north of Sedrun and from South to North south of Sedrun. The entire tunnel is made up of twin tunnels and connecting galleries and is 57km long. A total of four TBMs are being used to excavate the twin tunnels. Two 9.5m diameter TBMs were used...
in the north end to excavate the Amsteg and then the Erstfeld lots, and the other two 8.8m
diameter TBMs were used to excavate the Bodio and then the Faido lots. The lots providing data
for this research are the Amsteg lot (Figure 2.1.3) and the Bodio lot (Figure 2.1.4).

The 34 km long Lötschberg tunnel comprises a northern (Frutigen) and southern (Raron)
portal and three intermediate access adits: Mitholz, Ferden and Steg (Figure 2.1.5). Two 9.5m
diameter TBMs were used to excavate the Steg and Raron lots (Figure 2.1.6), both of which are
of interest to this research.

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**Figure 2.1.1:** Map of Western European rail network showing importance of Swiss railways (BLS
Alptransit, 2004); Lötschberg (left) and Gotthard (right) tunnel locations are highlighted.
Figure 2.1.2: Plan view of Gotthard Base tunnel. Worksites from which data were collected are Amsteg, a northern adit, and Bodio, at the South portal. (Alptransit, 2005).
Figure 2.1.3: Close up view of Amsteg worksite showing access adit, twin tunnels and connecting galleries (Alptransit, 2008).

Figure 2.1.4: Close up view of Bodio worksite showing access adit, twin tunnels and connecting galleries (Alptransit, 2008).
Figure 2.1.5: Plan view of Lötschberg Base tunnel. Worksites from which data were collected are Steg, a southern adit, and Raron, at the South portal (BLS Alptransit, 2008).

Figure 2.1.6: Close up of Steg (left) and Raron (right) worksites showing access adit, twin tunnels and connecting galleries (BLS Alptransit, 2008).
2.1.2 Difficulties Encountered in the Alpine Tunnels

Several excavation difficulties arose in the Alpine tunnels requiring research to understand the processes controlling the difficulties. The two main difficulties addressed by this research are low penetration rates and face instability. The low penetrations rates specifically being addressed are those arising from poor chipping conditions rather than utilisation or support requirements, neither of which are addressed by this research. The chipping process is understood to be the most efficient method of TBM excavation and involves creating chips between adjacent cutters by inducing tensile fractures through intact rock. This research aims to identify the geological characteristics responsible for poor chipping. This requires an understanding of the conditions under which chips are and are not generated.

Face instability, as addressed in this research, involves the generation of muck with large slabs of largely intact rock by stress induced spalling in the tunnel face. This is not instability arising from pre-existing joints in a blocky rock mass, which is not addressed in this research. This research aims to identify the stress and geological conditions that combine to induce face instability in massive, whether isotropic or foliated, rock and provide an understanding of the processes involved in generating face instability.
2.2 Tunnelling and State of Practice for TBM Design

2.2.1 Fundamentals of TBM Tunnel Excavation

2.2.1.1 TBM Types and Components

A visit through the Herrenknecht TBM manufacturing plant is used here to show the different aspects of a TBM. The final machining of TBM parts, as well as the assembly of all parts necessary for TBM testing with the client is conducted at the plant. Some portions of the TBM are assembled on-site, as they are not necessary for testing. Some TBM components are manufactured or refurbished at other Herrenknecht subsidiaries or partner companies.

The Valsugana TBM, in addition to several micro tunnelling machines and other unfinished large diameter projects, was completing its in-plant assembly for future testing with the client. This TBM is a single shield hard rock TBM, and is said to perform best in brittle rock, soft rock and varying formations. The excavation of the tunnel is achieved by roller face and gauge cutters installed on a flat, rotating head (Figure 2.2.1). The gauge cutters create a slightly larger opening than the TBM diameter, for backfilling of the liner behind the shield, and to break up muck in the invert. Torque for the rotation is achieved by hydraulic motors and thrust is achieved by hydraulic pistons (Figure 2.2.2) installed in the middle shield, that push on the completed segmental concrete liner after each stroke. On most Herrenknecht machines a stroke is approximately 2m of tunnel length and is governed by the extension length of the thrust pistons.

The liner is installed at the rear of the shield in segments that are brought along the length of the TBM by conveyors and are erected into place by a vacuum erector. The muck is collected from the tunnel face by peripheral buckets, dropped through muck chutes at the rear of the cutter head and carried along the length of the TBM by a conveyor belt (Figure 2.2.3). Ventilation is regulated using ventilation conduits along the length of the tunnel up to the TBM face (Figure 2.2.3).
Figure 2.2.1: Valsugana hard rock TBM cutter head (left) and close up of cutter (right, courtesy of M.S. Diederichs).

Figure 2.2.2: Hydraulic pistons on Valsugana TBM.
The cutting discs are installed in the cutter face in such a way that they may be changed from the rear of the cutter head (Figure 2.2.4) to minimise dangerous exposure of operators to the tunnel face. The direction of boring of the cutter head is accomplished with hydraulic pistons (Figure 2.2.5) positioned in such a way as to provide vertical and lateral thrust, a combination of which gives the TBM cutter head the possibility of performing any direction change. The final tunnel will be lined with concrete, which is backfilled during installation, and will be ready for installation of final components directly behind the TBM.

An open hard rock TBM (used in Amsteg, Bodio and Lötschberg) differs from the Valsugana TBM in that concrete segments are not installed during excavation. Instead, the rock is supported with temporary supports including rock bolts, wire mesh and steel arches (Figure 2.2.6), sometimes covered with a layer of shotcrete. Since the machine does not use a liner to provide thrust, the thrust is provided by two grippers, one on either side of the TBM, which press against the tunnel wall for traction. The forward thrust is then provided by hydraulic pistons connecting the grippers to the TBM head (Figure 2.2.7). The need for preliminary support and a solid tunnel wall for the grippers are the two practically most relevant differences between open and shielded (Valsugana) TBMs.
Figure 2.2.4: Valsugana TBM: rear of cutter face, showing cutters in background, conveyor rollers at top and hydraulic motor hoses at left.

Figure 2.2.5: One of several hydraulic pistons providing directional thrust to Valsugana TBM cutter head.
Figure 2.2.6: Photo of tunnel wall in open TBM (Amsteg) showing preliminary support: rock bolts, wire mesh and steel arches.

Figure 2.2.7: Open TBM gripper (Bodio) looking forward toward TBM head. Hydraulic piston in foreground advances gripper at beginning of stroke and advances TBM at end of stroke. Two hydraulic pistons in background provide thrust to TBM cutterhead.
The performance of a TBM is gauged by both the penetration rate, that is the distance the machine can penetrate into the rock per revolution of the cutter head with no consideration for other factors such as support, and the advance rate which is the distance excavated over a certain time frame taking into consideration all operation factors. The penetration rate reflects the behaviour of the rock at the tunnel face and the effectiveness of the TBM head design (disk type, size, spacing, etc.) for a particular rock. The advance rate reflects the penetration rate as well as the behaviour of the rock within the roof and walls of the tunnel and operational aspects such as material handling, trailer efficiency, etc.

2.2.1.2 TBM Development and Design Criteria

A larger cutter diameter results in a longer circumference and longer cutting length per revolution, therefore, increasing the cutter life. As a result, 432 mm and 483mm cutters are preferred for hard rock tunnelling. The larger the cutter, however, the greater its weight, and to facilitate cutter change 432 mm cutters are most often employed because they can be more easily handled by a single worker.

Back loading cutters are preferably used for projects where stability of the face is a concern, to assure that workers can perform cutter changes behind the cutter head protected from any falling rock blocks. Only in projects where tunnel face stability is not an issue, can front loading cutters be used as access to the tunnel face for cutter changes is required.

The cutter head shape may be flat, spherical or conical. Flat heads are used to reduce wear on cutters and cutter head from regrinding of excavated muck due to better muck flow into buckets, reduce potential for cutter damage from falling blocks and reduce the potential for face instability in cases of blocky ground by holding dislodged blocks in place.

Buckets can be located in the face or the periphery of the cutter head, or both. It has been found, however, that peripheral buckets are more effective at collecting muck from the invert than are face buckets at collecting muck as it falls along the face. As a result peripheral buckets are preferentially used in hard rock TBMs. Bucket grills are installed at the bucket opening to restrict the size of muck blocks entering the chute to reduce damage to the conveyor belt from excessively large and heavy rock blocks. The blocks that do not fit in the grills are broken up by the gauge cutters until the debris finally passes into the buckets and out of the tunnel.

The rotation of the cutter head can be provided by hydraulic motors or electrical motors. The electrical motors can provide a greater and continuously varying torque. Electric motors can also provide a higher maximum torque, required when starting up the cutter head. The torque will be maximised to overcome inertia, friction and blockage on an uneven rock face at start up,
and can be reduced once the cutter head begins rotating. The drawbacks of electric motors are the
cumbersome size of the motors, which must be located at the cutter head and the added cost.
Electric motors for hydraulic power can be located in the back-up of the TBM, where conditions
are not so crowded, but the energy efficiency is mostly used up in conversion of the electric
energy into hydraulic energy and transmission of the hydraulic fluid from the motors to the head.
Smaller hydraulic motors can then be located in the cutter head, and do not contribute to
crowding as much as electric motors. The motors are selected based on geometry, required
torque, TBM diameter, TBM type, rock type and cost.

The thrust is provided by cylinders pressing against the lining or grippers. Hard rock
TBMs, for situations where tunnel wall stability is expected to be good, can be equipped with
single (Bodio, Amsteg, Lötschberg) or double grippers. Single gripper machines are preferred by
Herrenknecht due to the added directional control available for the cutter head and increased
support installation (L1 area) area between the cutter head and single gripper. Fully lined tunnels,
in both hard and soft rock, can use cylinders pressing on the lining to provide thrust (Valsugana).
Double shield machines (Guadarrama) are outfitted with both grippers and pistons, and can be
operated by one or the other, depending on ground conditions. Microtunnelling machines may
obtain their thrust by pipe jacking, horizontal directional drilling or raise boring.

To minimise wear on various components of the cutter head high strength steel plates
inserts are installed in high wear areas. High strength alloy wear plates are welded to high wear
surfaces on the bucket lips and cutter face to act as sacrificial wear components, and the welding
is performed using high strength welding compounds. The cutter ring is made of ultra high
strength cutter steel, and is generally a trade secret of TBM manufacturers.

2.2.1.3 TBM Design Concepts Specific to Hard Rock Excavation

Larger cutters have a longer cutter life and can withstand a higher sustained cutter thrust,
and are therefore, preferentially used in hard rock TBMs. The longer cutter life results in fewer
cutter changes and shorter downtimes, while the higher cutter thrust can be translated into higher
penetration rates. Both of these factors lead to a higher advance rate and lower cost.

The average cutter spacing most often used for Herrenknecht machines is around 80mm.
The spacing affects the generation of chips. If the spacing is too wide then cracks either do not
interact and coalesce or it takes a large number of cutter passes for the cracks to interact and
produce chips. If the spacing is too narrow then small chips are created, reducing the productivity
of the boring process. The spacing also depends on the cutter thrust, since with higher cutter
thrust longer cracks can be generated and chips can be generated with fewer cutter passes for the same spacing. As a guideline, chips should be generated after 2-3 passes. The spacing should be determined for a certain thrust so that chips will be formed after 2-3 passes.

For hard rocks a large thrust is necessary for the cutter tip to penetrate into the rock as it passes. The penetration of each cutter tip results in a concentric crushed zone, called a kerf. The propagation of cracks away from the crushed zone, towards neighbouring kerfs, generates chips. Clearly, a greater penetration of the cutter into the rock results in better crack formation and earlier chip formation. In order to optimise the excavation, therefore, the thrust must be increased for hard rocks, meaning that the grippers or pistons must be capable of delivering the required thrust, and the cutters must be capable of handling the high loads.

Cutters may fail in a number of ways, depending on the cause of failure. Wear is the most common cause of cutter failure, and as such, is predicted in the design stage to determine cutter change scheduling during TBM operation. In some cases, impacts from rock blocks cause chipping of the cutter ring.

Normal wear is the wear on the cutter ring caused by the unhindered rolling along the tunnel face. It can manifest itself in a number of geometries depending on such factors as rock hardness, thrust and temperature. Uneven wear is the wear on only certain portions of the cutter ring due to bearing failure. The wear may be on one portion of the cutter ring only, if the bearing is completely jammed. It may also be on several portions of the cutter ring if the bearing is jammed and then forced to roll momentarily due to impact of a large block or a portion of an uneven rock face.

### 2.2.1.4 Hard rock cutting mechanisms

Many investigations into the hard rock cutting mechanisms have been performed and it is generally agreed that a cutter first creates a crushed zone at the cutter-rock interface (Figure 2.2.8). The stresses from the thrust of the cutter are transmitted through this crushed zone into the adjacent undamaged rock. The induced stresses cause fracturing of the rock away from the crushed zone. At some point cracks generated by cutter passes extend either to the rock surface or to cracks extending from adjoining kerfs to produce chips. This is referred to as the chipping process.
Chips account for the largest amount of rock removal, although their formation requires much less energy than the formation of the crushed zone. The formation of chips is, therefore, critical for optimum penetration of the TBM. Gertsch (2000) stated that the rolling force on a cutter is the most critical quantity in disc cutting because the rolling force ($F_r$ in Figure 2.2.9) is associated with and affected by the formation of the crushed zone, while the normal force ($F_N$ in Figure 2.2.9) is associated with chip and cutting formation.

Bruland (1998) states that fracturing in the rock helps ‘precondition’ the face so that boring is easier but it may also cause instabilities in the face. This can lead to fall-out of certain blocks, potentially leading to damage of the cutter head. The areas where fall out occurred will have gaps in the rock and cutters passing over these gaps will not transfer any thrust to the rock. This thrust will be distributed among those cutters that are in contact with the rock, leading to increased average and a more dynamic load situation on all cutters.

Some debate exists as to whether or not a radial tensile crack zone extends from the crushed zone. Gertsch (2000) states that radial cracks, if they exist, should be present in chips from linear cutting tests, but they are not. His work with rock blocks also suggests that median, or vent, cracks are not formed due to the self-confinement of the block samples.

Most discussions, however, (i.e. Bruland, 1998; Cigla et al., 2001; Zhang, 2001) describe the tensile crack formation as being radial from the crushed zone, and may or may not specifically
include median cracks. Differences in test geometry and rock type may account for the difference in opinion, and evidence from chips derived for the TBM may provide the best evidence. Bruland (1998) conducted TBM chip analyses and found evidence of fractures from previous passes on the chips. These fractures were interpreted as radial cracks and provide evidence for radial fracture networks being generated by repeated cutter passes.

The different forces on a cutter are represented in Figure 2.2.9. The thrust and rolling components of force, $F_n$ and $F_r$, respectively, arise from the thrust and torque applied by the cutterhead and resistance from the leading edge of the rock in the kerf, which has height $P$. The leading edge is due to the penetration depth achieved during each pass of the cutter and is collected from onboard computers as the penetration rate in mm/revolution. The total machine thrust and torque are not simply a summation of the thrust and rolling components of force on the cutter, but also arise from friction on the cutterhead and losses due to the imperfect stiffness of the system. The TBM clearly has design limitations in terms of maximum force on the cutters, beyond which damage can occur, as well as limitations on pressure provided by the hydraulics systems and limitations on mucking capacity of the conveyor belt. All of these limitations affect the operation of the TBM in different rock conditions.

TBM shield friction is thought to be a major contributor to the ‘other’ category of thrust force (contributors other than the thrust force on the cutters). Due to differences in TBM diameters and shield lengths, the friction on different machines would also be different. In an endeavour to obtain the most mutually comparable machine performance data as possible, the friction component of thrust could be identified and removed from total thrust. This procedure is described in Appendix B.1.
2.2.1.5  **Cutter Rolling Forces and Torque**

While the importance of thrust is evident from the chipping discussion, the importance of torque is less evident. Several authors (Samuel & Seow, 1984; Rostami, 1997) have related torque to the rolling force on cutters, where the rolling force is the tangential force acting on the cutter during excavation. The rotational force can be estimated by a number of methods (for example: Roxborough & Phillips, 1975; Samuel & Seow, 1984; Sanio, 1985; Rostami, 1997) based on physical characteristics of the cutters and their arrangement on the TBM head.

Two general types of torque can be defined: global torque and friction torque. The global torque is the torque required to make the TBM head rotate, and takes into account all internal and external rotational forces. Samuel & Seow (1984) do not specify the individual contributions to the global torque, whereas Rostami (1997) states that the sum of the friction torque (cutterhead torque requirement) and the rotational speed of the cutterhead is used to calculate the cutterhead power requirements (indicating general torque).

The friction torque is the torque required to make the cutters roll on the rock surface, in other words the magnitude of torque exerted on the tunnel face (Samuel & Seow, 1984). The friction torque will be the sum of the torque contributions of each cutter, calculated as the sum of
the product of the individual rolling forces and their radial distances away from the cutterhead centre (Rostami, 1997). In Samuel & Seow (1984) all rolling forces are assumed to be equal, while in Rostami (1997) the rolling forces can be different, depending on the cutter geometry and location.

These show that the general torque value collected from the TBM driving data represents inputs from different systems within the TBM cutterhead. Examination of the TBM torque data and evaluation of the behaviour of the torque with respect to thrust and penetration during actual TBM driving in hard rock has led to many questions and indicators to the importance of torque during driving, and in interpretation of TBM driving data.

Figure 2.2.10 shows a phenomenon observed during the slow increase in thrust applied to the TBM operated in the Leventina gneiss of the Bodio tunnel. A penetration rate plateau was identified in which the thrust increases with little or no increase, or even decrease, in penetration (Figure 2.2.11) while the torque tends to oscillate and shows a moderate increase with little relative increase in penetration (Figure 2.2.12). Figures 2.2.13 and 2.2.14 show the thrust and torque values for the plateau of Figure 2.2.10 in a different stroke in Leventina gneiss. The torque in Figure 2.2.14 oscillates but does not increase substantially during the plateau.

![Figure 2.2.10: Graph of penetration rate versus gross thrust for low thrust portion of a single TBM stroke in Leventina gneiss. (Bodio stroke P0869)](image-url)
Figure 2.2.11: Gross thrust and penetration rate values over the distance for which the penetration rate is steady for data in Figure 2.2.10. (Bodio stroke P0869)

Figure 2.212: Torque and penetration rate values over the distance for which the penetration rate is steady for data in Figure 2.2.10. (Bodio stroke P0869)
Figure 2.2.13: Gross thrust and penetration rate values over the distance for which the penetration rate is steady. (Bodio stroke P0877)

Figure 2.2.14: Torque and penetration rate values over the distance for which the penetration rate is steady (Bodio stroke P0877)
Theoretical formulations for the relationship between thrust, torque and penetration (Okubo, Fukui and Chen, 2003) have not predicted these observations. The methodology presented in Rostami (1997) consists of first determining the average pressure bulb resulting at the cutter/rock interface during excavation at a particular depth, translating that into a total force, which is then broken into the normal and rolling forces and converted into total thrust and torque, as follows (see Figure 2.2.9):

\[ F_N = TR\Phi P_r \]  
\[ F_R = F_n RC \]  
\[ P_r = C\sqrt{\frac{S\sigma_c^2\sigma_t}{\Phi\sqrt{RT}}} \]  
\[ RC = Tan\left(\frac{\Phi}{2}\right) \]  
\[ \Phi = \cos^{-1}(1 - \frac{p}{R}) \]

where \( F_N \) and \( F_R \) are the normal and rolling forces, respectively, \( T \) is the cutter tip width, \( R \) is the cutter radius, \( C \) is an empirical constant and \( = 2.12 \), \( S \) is the cutter spacing, \( \sigma_c \) and \( \sigma_t \) are the UCS and tensile strength, respectively, and \( p \) is the penetration depth. The equation for \( P_r \) is based on imperial units, therefore all factors must first be converted to imperial, and the \( P_r \) is returned as psi, which can be converted to MPa for determination of the normal and rolling forces in kN. An attempt was made to determine the pressure from physical first principles but the complexity was too great and a multivariate regression analysis was used on the Colorado School of Mines linear cutting test database instead (Gertsch, 2000). This provides a good correlation to the dataset but the predictive capabilities are still limited (Gertsch, 2000).

Total thrust is the product of the number of cutters and the normal force:

\[ F = nF_N \]  

Total torque relates the sum of the rolling forces multiplied by the average moment arm of the TBM head (where \( D \) is the distance from the cutter to the centre of the cutter head):

\[ T = nF_R DO.3 \]

A comparison of the theoretical thrust and torque values calculated from Equations 2.2.6 and 2.2.7 and average TBM stroke data from the Leventina gneiss has shown that the trends are similar, except at high penetration rates limited by other TBM factors, although the values are not exact (Figures 2.2.15 and 2.2.16). This is likely due to incorrect input parameters for Equations 2.2.6 and 2.2.7 since UCS and \( \sigma_t \) are not known for the Leventina gneiss for which the TBM data were collected, as well as lack of information regarding energy losses in the TBM system.
Figure 2.2.15: Real and theoretical (Rostami, 1997) thrust versus penetration graph for average Leventina gneiss TBM data.

Figure 2.2.16: Real and theoretical (Rostami, 1997) torque versus penetration graph for average Leventina gneiss TBM data.
Figures 2.2.17 and 2.2.18 show the results of a similar analysis undertaken with slow thrust increase data similar to Figure 2.2.10. The trends fit relatively well, with the differences arising from the same lack of exact input values as for the average TBM data. The slope for the torque-penetration graph is much higher with theoretical values than with real values, likely due to inequalities in cutter penetration over the TBM head. The data recorded by the TBM is average depth over the 58 cutters, but the torque represents the cumulative torque contribution from each individual cutter. An improvement to the comparison of the two datasets (Figure 2.2.19) was made by calculating the theoretical torque calculating the torque contributions from each cutter at varying depths, some cutters between 0.8 and 1.2mm and some cutters between 4.3 and 4.7mm to simulate the impact of some cutters penetrating deeper into the face than others, and summing the torque contributions from each cutter to determine gross torque. Although the fit is improved, it is somewhat artificial because losses are still not accounted for.

![Figure 2.2.17: Real and theoretical (Rostami, 1997) thrust versus penetration graph for slow thrust increase stroke in Leventina gneiss.](image-url)
Figure 2.2.18: Real and theoretical (Rostami, 1997) torque versus penetration graph for slow thrust increase stroke in Leventina gneiss.

Figure 2.2.19: Real and theoretical (Rostami, 1997) torque with varied penetration depth versus penetration graph for slow thrust increase stroke in Leventina gneiss.
What this analysis shows is that theoretical formulations cannot alone be used to predict the cutter excavation process. The inflection point in the thrust-penetration graph in Figure 2.2.10 is interpreted to represent the point at which the cutters cease to simply grind the rock and begin to induce fractures that lead to chipping. This inflection point is partially simulated in the theoretical formulation by Rostami (1997) with its power function shape, but it does not adequately describe the grinding plateau and distinct change from grinding to chipping seen in the TBM data. In addition, the torque formulation only partly fits the TBM data in that it is also a nearly linear function. The improvement made by adding variable cutter depths to the function suggests that cutters do not necessarily all excavate at the same depth and using average kerf depth is an oversimplification of the cutter excavation process. This is especially true during chipping, which is a heterogeneous process leading to different kerf depths depending on chip status adjacent to cutters.

2.2.1.6 Face Strength and Behaviour Estimation

Okubo et al. (2003) discuss a variety of methods for determining the rolling and normal forces on the TBM cutter. By determining some estimate of the geological conditions, the rolling and normal forces can be calculated and used to assess the thrust and torque requirements during TBM design. By using this principle in reverse, Fukui and Okubo (2005) showed that the rock strength can be derived by measuring the thrust and torque required to penetrate to a particular depth, using the following equations:

\[ \sigma_c = \frac{F}{(c_1 p)} \]  \hspace{1cm} 2.2.8

\[ \sigma_c = \frac{T}{(c_2 p + 1)} \]  \hspace{1cm} 2.2.9

where \( \sigma_c \) is the UCS, \( F \) is the total thrust force, \( T \) is the total torque, \( p \) is the depth of penetration and \( c_1 \) and \( c_2 \) are constants derived by regression of previous TBM projects (Fukui and Okubo, 2003).

The development of \( c_1 \) and \( c_2 \) is described in Fukui and Okubo (2003), and are used to account for the specifications of TBM used for different projects as follows:

\[ c_1 = n(4.37d - 1.43) \]  \hspace{1cm} 2.2.10

\[ c_2 = c_1(2.19 - 0.1D) \]  \hspace{1cm} 2.2.11

where \( n \) is the number of cutters, \( d \) is their diameter distance to the centre of the cutter head and \( D \) is the TBM diameter. These values were determined by regression analysis and are not based on a mechanical analysis of the cutter geometry, such as the ones presented in Rostami and Ozdemir (1993) and Sanio (1985), for example.
Rostami and Ozdemir (1993) derived a mechanical analysis of the rolling and normal forces on a cutter using a geometrical analysis of the force vectors as a function of penetration depth. Linear cutting tests on individual cutters were analysed within this framework and a regression analysis of the data resulted in a relationship between the TBM specifications and geological conditions for an analysis of the required thrust and torque capabilities for TBM design. This methodology was improved in Rostami (1997), and the equations presented therein were used to replicate the tunnel face strength estimation method by Fukui and Okubo (2005).

Sanio (1985) and Nelson (2003) both use the relationship

\[ F_s = F_s C \sqrt{\frac{P}{d}} \]  

where C is 0.8 or 0.9 according to Sanio (1985) and Nelson (2003), respectively. By deriving the normal force using the method by Rostami (1997) or Fukui and Okubo (2003), the rolling force can be estimated according to Sanio (1985) or Nelson (2003).

By reversing each of the methodologies described, including the one by Rostami (1997) in Section 2.2.1.5, the face strength could be estimated, as suggested by Fukui and Okubo (2005). In order to investigate how well this works for the TBMs used in this research several formulations of the methodologies described above were used to calculate the total thrust and torque on TBM S-229, as shown in Table 2.2.1. Okubo et al. (2003) provide a chronological summary of various published methods for estimating normal and rolling force. Of these, the methods outlined by Sanio et al. (1985), Nelson et al. (1985) and Rostami and Ozdemir (1993) were found to be most relevant (Gertsch, 2000; Okubo et al., 2003). The estimates were compared to real average data for a single stroke in granitic gneiss whose strength is estimated at UCS = 200MPa, tensile strength = 8MPa and penetration depth = 2.819mm. A description of each method is as follows:

1. Data: as recorded by onboard TBM computer. This data includes friction, loaded bucket weight, inertia from machine weight.
2. Average forces: actual calculation using method found in Rostami and Ozdemir (1993) and TBM parameters from S-229. Average normal and rolling forces determined and used to calculate Thrust and Torque.
3. Individual forces: modified method from Rostami and Ozdemir (1993) applied to each individual cutter. Takes into account reduced penetration at gauge cutters, which reduces rolling force, due to their angled orientation to the tunnel face. Normal and rolling forces are calculated for each cutter, summed and used to calculate Thrust and Torque according to Rostami and Ozdemir (1993).
4. Individual forces (includes rolling forces): modified method from Rostami and Ozdemir (1993) applied to each individual cutter. Takes into account reduced penetration at gauge cutters, which reduces rolling force, as well as reduced torque at gauge cutters, due to their angled orientation to the tunnel face. Torque is calculated for each cutter, then summed to determine total Torque. Thrust is determined as for Method 3.

6. Individual forces: equations and constants from Fukui and Okubo (2003) derived to the individual cutter scale and summed to determine total Thrust and Torque. Torque takes into account reduced penetration depth at the gauge by using depth for each cutter and $c_2$ is calculated for each cutter.

7. Individual forces (including rolling forces): normal force estimate derived by method presented in Fukui and Okubo (2003) and used to determine rolling forces using methodology presented in Rostami and Ozdemir (1993), which are then used to calculate individual cutter torque, as in Method 4.

8. Individual forces (including rolling forces): normal force estimate derived by method presented in Fukui and Okubo (2003) and used to determine rolling forces using methodology presented in Sanio (1985), which is similar to the method presented in Nelson et al. (1985), and are then used to calculate individual cutter torque, as in Method 4.

Methods 2 and 5 are the direct application of the methods as they are published. These methods average the forces and are only an approximation of the actual thrust and torque. In order to improve their comparison to data, an attempt was made to determine the actual forces on each cutter by using real cutter spacing, cutting depth and angle to the tunnel face. Different iterations of modification were used to determine whether or not they improved the estimates.

Table 2.2.1: Estimation of total thrust and torque

<table>
<thead>
<tr>
<th>Method</th>
<th>Normal Force</th>
<th>Rolling Force</th>
<th>Total Thrust</th>
<th>Total Torque</th>
<th>T/Fp^0.5</th>
<th>T/F</th>
<th>Source</th>
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</thead>
<tbody>
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<td>1. Data</td>
<td>None</td>
<td>None</td>
<td>16481</td>
<td>1930</td>
<td>2.21</td>
<td>0.12</td>
<td>Gotthard</td>
</tr>
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<td>2. Average Forces</td>
<td>188</td>
<td>15</td>
<td>11658</td>
<td>1358</td>
<td>2.18</td>
<td>0.12</td>
<td>1</td>
</tr>
<tr>
<td>3. Individual Forces</td>
<td>Individual</td>
<td>Individual</td>
<td>11243</td>
<td>1285</td>
<td>2.07</td>
<td>0.11</td>
<td>1</td>
</tr>
<tr>
<td>4. Individual Forces (incl. rolling)</td>
<td>Individual</td>
<td>Individual</td>
<td>11243</td>
<td>2144</td>
<td>3.6</td>
<td>0.19</td>
<td>1</td>
</tr>
<tr>
<td>5. Direct Calculation</td>
<td>None</td>
<td>None</td>
<td>16006</td>
<td>1048</td>
<td>1.24</td>
<td>0.066</td>
<td>2</td>
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<tr>
<td>6. Individual Forces</td>
<td>Individual</td>
<td>Individual</td>
<td>15004</td>
<td>2175</td>
<td>2.73</td>
<td>0.14</td>
<td>3</td>
</tr>
<tr>
<td>7. Individual Forces (incl. rolling)</td>
<td>Individual</td>
<td>Individual</td>
<td>15004</td>
<td>2907</td>
<td>3.65</td>
<td>0.19</td>
<td>Combined 1 and 3</td>
</tr>
<tr>
<td>8. Individual Forces (incl. rolling)</td>
<td>Individual</td>
<td>Individual</td>
<td>15004</td>
<td>3278</td>
<td>3.96</td>
<td>0.21</td>
<td>Combined 3 and 4 or 5</td>
</tr>
</tbody>
</table>

1 (Rostami and Ozdemir, 1993)  
2 (Fukui and Okubo, 2005)  
3 (Fukui and Okubo, 2003)  
4 (Sanio, 1985)  
5 (Okubo, Fukui and Chen, 2003)
Fukui and Okubo (2005) state that the ratio \( \frac{T}{Fp^{0.5}} \) should remain constant for all machines. When comparing the estimates to the data not only were the values of thrust and torque compared, but their ratio with respect to penetration depth was also compared. The original method from Rostami (1997) (method 2) and the first modified version from Fukui and Okubo (2003) (method 6) resulted in the closest ratio estimates, although the latter had the best estimates for total thrust and torque.

Regardless of the proximity of the results, no single method is capable of accurately predicting thrust and torque for the following reasons: friction on the shield, cutter head and cutters, as well as added torque due to inertia and weight of muck in buckets are indirectly accounted for during the regression analysis and included in the derivation of the constants used in all methods. The particular effects of these parameters on S-229 is not necessarily accounted for exactly by these methodologies, allowing errors to arise. Since it is not possible to completely bring these methodologies to first principles, they remain a broad estimate and any calculation of face strength by the method suggested by Fukui and Okubo (2005) can only be used as such.

The principle that the ratio between torque, thrust and penetration rate are the same for a particular machine is useful for analysing TBM machine data in less than optimal conditions. The ratio should only hold true if the machine is running at capacity, that is, either thrust or penetration rate limited, since it is only under these conditions that penetration rate and thrust are related by rock mechanics, and not external factors such as torque limits. Figure 2.2.20 shows the values of the ratio for the S-229 TBM based on average stroke data in varying geological conditions. The average value of the ratio over this interval is 2.32. The same machine has an average ratio of 1.82 over a different interval under different geological conditions, while a very similar machine has a ratio of 2.09 in a parallel tunnel under slightly different conditions.

An analysis of the components of the ratio can help identify the most correct ratio range, which can then be compared to TBM performance when the ratio deviates from the most correct range. Increased torque or decreased thrust and/or penetration rate (depth) will lead to an increased ratio, with the converse being true. Situations where torque has not reached its maximum limit are used to determine the most correct range.
The drillability index (DI, which is the ratio of penetration rate to thrust, similar to the inverse of equation 1) is used to examine the normalised penetration rate according to thrust. This value provides information concerning the face strength and condition during TBM excavation, as suggested by Fukui and Okubo (2005). The net advance rate (NAR, which is the ratio between the distance covered in a stroke and the total active driving time required to cover the distance) is a measure of the impact of unfavourable geological conditions to advance, for example very tough rock or face instability. In favourable geological conditions NAR and speed should follow a 1:1 relationship, as seen in Figure 2.2.21.

Figure 2.2.22 shows the NAR versus DI data from TBM S-210. The relationship between the NAR and DI is roughly linear, suggesting that in this rock type the thrust or penetration rate were at their maximum and the resulting advance rate was related to the rock mechanics process involved in chip formation. As discussed earlier, the data in this dataset displays a relationship controlled by rock mechanics and can, therefore, be used to explore the relationship between thrust, penetration rate and torque, according to the methodology outlined in Fukui and Okubo (2005).
Gertsch (2000) states that two flaws exist when attempting to use empirical relationships for site-specific prediction:

- The prediction can be weakened in newly-encountered, untested rock, since the relationship is based on a database of known and tested rocks.
- While the regression curve may have a high correlation coefficient to a large dataset, it does not account for scatter and can result in misleading predictions when trying to predict performance for a specific case.

Figure 2.2.21: TBM performance data for S-210 showing a direct relationship between NAR and Speed, suggesting favourable geological conditions for TBM excavation
Figure 2.2.22: TBM performance data for S-210 in tough rock suggests a rock mechanical relationship between thrust and penetration (and advance rate) in this rock type.

A limitation exists for the back analysis for a full-faced machine: penetration rate is for the entire machine, related to the depth of breakage (grinding, chipping and kerf creation), and not necessarily to the depth of penetration of the cutters (kerfs only). Based on the trials regarding theoretical formulations and rock strength back-calculation in this Section and Section 2.2.1.5, the limitations in these formulations and the difficulty in obtaining comparable theoretical and real values for various TBM performance parameters, it is suggested that these formulations are not capable of capturing the complexity of the cutter excavation process, especially with respect to the chipping process.

2.2.2 Rock Behaviour and TBM Performance

2.2.2.1 Introduction to Rock Behaviour during TBM Excavation

The cutter excavation process introduced in Section 2.2.1 comprises the removal of material at the tunnel face through grinding or chipping induced by cutter thrust against the rock.
This progressive excavation technique results in specific tunnel wall and face stability issues arising from the smooth excavation boundary, stress rotation at the tunnel face (Diederichs et al., 2004) and the lack of induced stress release through the preconditioning of the rock mass that occurs during drill and blast excavation.

Stresses induced by the tunnel excavation, especially at high overburden, can exceed the strength of the intact rock and lead to spalling in the tunnel walls. Diederichs et al. (2004 a) state that in smoothly bored tunnels, damage at the excavation boundary is often in the form of spalling and notch formation (Figure 2.2.23). Spalling in bored tunnels, in general, cannot be prevented once the induced stress exceeds the excavation boundary strength; however, failed material can be held in place by support, thus reducing the potential for damage to the TBM and worker injuries from falling material and controlling further spalling into the rock. This type of failure is typically more of a nuisance (Hoek, 2004) rather than a severe danger and can lead to lower TBM advance rates when time requirements for support and/or cleanup of fallen material exceeds the time required for the TBM to complete one stroke, or if support cannot be installed while advancing.

In some cases the induced stresses, rock strength and any structures or fabric, are such that alone or in combination with each other they create conditions where rock bursts (violent failure of the rock) can occur (Figure 2.2.24). Rock bursts can pose a severe threat to machine and workers since energy stored in the rock is released during failure. Kaiser et al. (1996) suggest that support function is not to reinforce the rock, because rock bolts do not apply sufficient stress to prevent failure, but rather to keep the broken rock in place once it has failed. If this can be done, then the safety of the work area and the utilisation of the machine are more likely to be preserved.

Diederichs et al. (2004 b) have shown that rotation of the excavation induced stresses due to the progressive advance of a deep bored tunnel can dramatically affect the behaviour of the tunnel walls. They show that the stress rotation ahead of an advancing face can result in induced stresses that exceed the damage initiation threshold of a rock, inducing damage that can persist in the tunnel walls once the face has advanced. In such situations the damage may vary significantly along a tunnel path and support requirements may have to be adapted to the damaged behaviour of the rock. The extent and location of such damage depends largely on variations in foliation and fabric, mineralogy, microtectonic characteristics, etc.
A typical type of rock behaviour during excavation is fall of ground arising from the intersection of pre-existing fractures and joints with each other and the excavation boundary to produce blocks that are released by gravity fall (Figure 2.2.25). This type of failure can be very problematic and blocks must either be removed or supported with ground support, such as rock bolts, cable bolts, mesh and shotcrete. If this type of failure occurs in the tunnel face (Figure 2.2.26) this can lead to problems in the buckets and conveyance system. Thorough mapping of joint sets in the rock mass can greatly aid in predicting this type of failure.

A less common type of rock behaviour encountered during TBM tunnelling is stress-related failure activated by foliation. Stress induced shearing was observed at the excavation boundary in Altkristallin in which massive, stable foliated rock took on a highly-weathered appearance due to shearing along the weaker foliation planes due to the in-situ stress at the tunnel.
boundary (Figure 2.2.27). Spalling and shearing along foliation planes can negatively impact tunnelling if it leads to the release of blocks defined by induced shear and extensile fractures, in particular when this occurs behind the shield or at the tunnel face (Figure 2.2.28). This type of failure can be supported in the tunnel walls in a similar fashion as for gravity falls, but can lead to severe bucket and conveyance problem when they occur at the tunnel face. Unlike gravity falls, stress-induced instability is much harder to predict, and improving this is a major goal of this thesis.

Figure 2.2.25: Photo of wedge failure in tunnel roof (left) with three-dimensional schematic of the failure (right) showing (clockwise from top left) wedge in plan view, oblique view, longitudinal view and cross-sectional view.

Figure 2.2.26: Schematic of wedge or block failure in tunnel face.
Figure 2.2.27: Photo of stress-induced shear on foliation planes resulting in ‘weathered’ appearance of otherwise stable tunnel wall.

Figure 2.2.28: Photo of tunnel face (left) with close-up of lower right failed section (right) showing possible structure ($S_1$ and $S_2$ are joints, $k_1$ is fabric) directions with respect to stress-induced failure.

Spalling in the tunnel face has also been observed in the massive crystalline rock of the Central Aar granite due to high overburden leading to extension into the tunnel axis, in particular in the centre of the face (Weh and Bertholet, 2005). This spalling behaviour impacted the TBM excavation in a similar manner as the gravity falls and foliation spalling and shearing.

### 2.2.2.2 Rock Mass Classification for Support

Three main systems are used for rock mass classification for excavation boundary support: Q (Barton et al., 1974), Rock Mass Rating (RMR) (Bieniawski, 1989) and GSI (Hoek, 1994; Cai et al., 2004). Q and RMR are based on quantified classification of discontinuities,
including length, aperture, spacing, weathering, roughness, water, etc. In addition, Q includes the stress/strength ratio and RMR includes the rock strength and discontinuity orientation with respect to the excavation. Both of these methodologies provide classification values that are commonly used in empirical systems and charts comparing such excavation details as unsupported stand-up time, support requirements and opening dimensions. In addition, they are used to select the method of excavation. GSI is a qualitative methodology for rock mass description based on the type of structure or discontinuities in terms of block interlock and the surface conditions of the discontinuities. The methodology also provides values that can be used to estimate the rock mass strength, given several intact rock strength parameters (Hoek, 1994). These classification schemes are used mostly for rock masses in which discontinuities play a large role in the rock mass strength and behaviour. This research addresses massive rocks in which the discontinuities play a minor role, although all rocks under investigation were classified by Q, RMR and GSI to ensure that they fall within the massive rock class.

What is interesting about the classification schemes presented here is the way in which the individual factors are combined: Q factors are multiplied and divided, RMR factors are summed and GSI factors are combined graphically. Each has strengths and weaknesses, which impact the way in which they are applied. The multiplication and division of factors in the Q factor means that different factors carry more weight than others depending on their use as a dividend or divisor. For example, very low values for divisors may result in very high quotients, which then impact the Q classification value significantly, which must be considered when assigning weightings to different factors. This is useful when some factors are considerably more important than others, and the multiplication of factors is useful when the final value is modified by each subsequent factor. The addition and subtraction of factors in RMR means that all factors are weighted equally and the factor values are large or small depending on the weight it should carry in the final RMR classification value. This system is more straightforward, in particular when the value is a representation of the cumulative impact of each factor. The GSI graphical methodology is very straightforward to use, but very complex to design, requiring large numbers of empirical situations on which the classification chart is based. The magnitude of values associated with the classification can be arbitrarily selected depending on the intended use, and their relative changes from one condition to the other are selected based on in depth examination of the relative impact of different rock masses on behaviour and strength.
2.2.2.3 TBM Operation

Large diameter, hard rock Herrenknecht TBMs are operated by an operator in an on-board cabin. The TBM is connected to a data acquisition system that monitors the status of a variety of components of the equipment in real time. Some TBMs are also equipped with digital video cameras of the muck on the conveyor belt. The TBM is operated based on the feedback displayed to the data acquisition system. The data that are most relevant to this research are penetration rate (mm/rev), speed (mm/min), gross thrust (kN), torque (% of max in MN.m), forces on the shield, conveyor belt speed, and visual monitoring of the conveyor belt. In addition, the operator typically can tell by the type of noise and vibration being felt on the machine what the excavation conditions are.

Three excavation scenarios are of importance to this research:

- Tough rock, which requires the application of the maximum thrust.
- Face instability, which causes a dramatic increase in both torque and material on the conveyor belt.
- Tunnel wall instability, which requires extra time to stabilise and can lead to delays in the stroke, affecting utilisation, which is not in the scope of this thesis.

Tough rock leads to low penetration rates and a variety of types of damage to cutters. Areas with exceptionally low penetration rate to thrust ratio are considered areas of tough rock. In this situation, the operator will typically attempt to hold the machine thrust at or slightly above the design maximum in order to increase the penetration rate.

Face instability leads to problems with conveyor belt capacity, damage to the conveyor, cutters and TBM cutterhead and increased torque. The most reliable indication of face instability is the digital video that monitors the muck on the conveyor belt, where increased block size in the muck corresponds to rapid increases in torque. In this situation the operator will cease to thrust while continuing to rotate the cutterhead and remove muck from both the cutterhead and the tunnel face. Once the torque has been lowered and the majority of the material has been removed (as seen on the digital video), the operator will resume excavation by again applying thrust. In this situation the penetration rate to thrust ratio may be high, but the total excavation time will be longer due to the need for frequent stops to clear the failed material in the tunnel face.

Tunnel wall instability can delay a stroke if the time to stabilise the tunnel walls is longer than the time required to excavate the rock. In this situation the operator may apply the thrust necessary to achieve the maximum speed (controlled mostly by conveyor belt capacity) or the
operator may need to lower the speed such that the tunnel wall can be stabilised as soon as it is exposed from behind the shield. In this situation, the thrust and penetration rate will both be low.

2.2.3 TBM Performance Prediction State of Practice

2.2.3.1 Introduction

There are two main TBM performance prediction models currently used in the industry, namely the model developed at the Colorado School of Mines (CSM) and the model developed at the Norwegian Institute for Technology (NTH) at the University of Trondheim in Norway (Rostami et al, 1996), and two lesser known methods: QTBM and an approach using the Rock Mass Rating classification system. Most tender documents include rock testing information used by these methodologies, including Cerchar, UCS, Brazilian tensile strength, and some rock mass classification. Additional strength tests, such as shear strength and directional strength testing of foliated samples is also common. In addition to these tests, each methodology used to design TBMs requires a series of method-specific tests and characterisations, described below.

2.2.3.2 NTH Model

The NTH model is an empirical model based on TBM performance data initially from Norway and is continuously upgraded as projects in Norway and around the world are completed. Rostami et al (1996) state that this model has good merits, is proven to be reliable and is the more widely used empirical model in industry, especially in Europe. The prediction model is described in University of Trondheim (1988a), a document that is updated periodically with additional tunnelling data.

The NTH model incorporates rock properties obtained from specially designed tests and rock mass properties obtained from a specially designed rock mass classification scheme to determine the net advance rate (also known as the penetration rate) and the cutter wear. The NTH documentation can also be used to determine the gross advance rate (also known simply as the advance rate), estimate project costs and design ground support and transportation in the tunnel. The latter sections are not covered in this review but can be found in University of Trondheim (1988a).

The net advance rate and cutter wear are calculated with input from both rock parameters and machine factors. The net advance rate rock parameters are based on structural mapping and classification, the drillability, represented by the drillability index (DRI), and the abrasiveness.
These are covered in some detail in the following sections. The machine factors include: cutter properties, cutter RPM, number of cutters, and installed effect and thrust (University of Trondheim, 1988a), but will not be discussed in this review. The cutter wear rock parameters include the cutter life index (CLI) and the mineral content, including quartz, mica and calcite, and amphibole (hornblende).

Rock mass fractures are also an important consideration in the NTH model and affect the net advance rate depending on rock mass and machine factors. The rock mass factors include planes of weakness (type, frequency and orientation) and the rock drillability.

2.2.3.2.1 Drilling Rate Index

The Drilling Rate Index (DRI) is calculated with the parameters determined by two specially designed laboratory tests. The first is the Brittleness Test, which is used to determine the brittleness value after 20 impacts, $S_{20}$, and the second is the Sievers’ Miniature Drill Test, which is used to determine the Sievers’ J value, SJ.

The two parameters are then located on a graph (F1) from which the Drilling Rate Index can be determined. University of Trondheim (1979b) states that the Drilling Rate Index can be seen as the rock’s Brittleness Value corrected for its Sievers’ J value. A relationship between DRI and UCS for different rock types is shown in University of Trondheim (1988a, p. 120).

The DRI value is then combined with the critical thrust per cutter and a fracturing factor ($M_0$), whose combination is then modified by a cutter size correction ($k_d$) and a cutter spacing correction ($k_s$).

$$M_1 = M_0 \cdot k_d \cdot k_s$$  \hspace{1cm} 2.2.13

This product, $M_1$ the critical thrust (corrected), is used to determine the penetration per revolution ($i$).

$$i = (M_B/M_1)^b$$  \hspace{1cm} 2.2.14

where $M_B$ is the gross thrust and $b$ is the penetration coefficient, a relationship between the critical thrust and the cutter diameter. The penetration per revolution is then combined with the cutterhead RPM (the result of the combination of several machine factors) to determine the net advance rate. The net advance rate is used to determine the gross advance rate, in terms of metres per week, both as a factor for utilisation and as the factor being corrected by the utilisation to determine the gross advance rate.
2.2.3.2.2 Cutter Life Index

The Cutter Life Index (CLI) is based on the Sievers’ J value and the Abrasion Value Steel (AVS), according to the following formula (University of Trondheim, 1988a):

\[ \text{CLI} = 13.84 \left( \frac{\text{SJ}}{\text{AVS}} \right)^{0.3847} \]

The CLI is combined with the cutter diameter to determine the cutter disc life, \( H \), in hours. This cutter disc life is then modified by correction factors for TBM diameter, cutterhead RPM, number of discs, and mineral content (quartz, mica and amphibole). These correction factors are used to determine the cutter disc life in terms of \( h/\text{cutter}, m/\text{cutter} \) or \( \text{sm}^3/\text{cutter} \). The latter two formulations require the net advance rate, determined as described above. The cutter disc life is used in the formulation for utilisation, which is then used to determine the gross advance rate.

2.2.3.2.3 Brittleness (Friability) Test

This test is based on the Swedish Brittleness Test. This test measures the percentage of rock that previously, passing through a 16.0 mm mesh, was retained on an 11.2 mm mesh and now passes through that mesh after a number of blows. This test measures the resistance of the rock to crushing from repeated impacts (Ozdemir and Nilsen, 1999).

In the Norwegian version 500g of the screened rock aggregate, with a density of 2.65 g/cm\(^3\), is loaded into a mortar and is crushed by 20 drops of the weight from a height of 250 mm. The impact mass, or pestle does not directly impact the rock. Rather, the pestle rests on the rock and is itself hit by a falling 14 kg weight, transferring the energy of the falling mass to the rock in the mortar (Bamford, 1984).

The sample weight is corrected to obtain a constant volume if the density of the rock is not 2.65 g/cm\(^3\). The brittleness value is the percentage of the rock that passes through the 11.2 mm mesh, and is taken as the mean value of at least 3-4 parallel tests (Bamford, 1984; University of Trondheim, 1979b).

2.2.3.2.4 Sievers’ Miniature Drill Test

The miniature drill test is performed on a pre-cut rock sample. The Sievers’ J Value, SJ, is the depth of the drill hole, in 1/10 mm, after 200 revolutions of the miniature drill, taken as the mean of 4-8 drill holes. The bit is sharpened after every test to ensure a constant geometry (Bamford, 1984; University of Trondheim, 1979b; Ozdemir and Nilsen, 1999).
The cut surface of the rock must be oriented perpendicular to the foliation and the SJ value parallel to the foliation is usually the one used to determine DRI. If the SJ measured perpendicular to foliation is different that the SJ measured parallel to foliation, then this may show that penetration rates are dependent on the angle of the excavation to the foliation. There are a few limitations and caveats to this test:

- Only the drillability is measured in this test, not the wear number or the abrasiveness. Bamford (1984) states that neglecting to measure half of the available information seems like a major omission.
- University of Trondheim (1979b) states that this test is sensitive to minor quartz lenses that may be found in the rock sample, and as a result the Sievers’ J value is not a good measure of anisotropy.

2.2.3.2.5 **Abrasion Value Test**

University of Trondheim (1979b) defines the Abrasion Value, AV, as “a measure for rock ‘powder’ abrasiveness on a tungsten carbide test bit.” A similar test on steel is also performed to provide information for steel TBM cutter discs.

Bamford (1984) describes the test piece as being a rectangular prism of tungsten carbide, which is 30mm long and 10 mm wide, with a curved wearing face that has a 15 mm radius of curvature. The abrasive rock powder is crushed rock with a diameter < 1 mm, simulating the drill cutting fines. The powder is fed onto a rotating steel disc, onto which the test bit is pressed by the 10 kg weight. The steel disc is rotated a total of 100 times in a 5 minute time span. The Abrasion Value is the mass of material lost, in mg, from the tungsten carbide test bit at the end of the test. The test bit is carefully ground after every test according to specific procedures (Ozdemir and Nilsen, 1999).

The Abrasion Value Steel is performed on the same apparatus, however, the test bit is made of TBM cutter steel and the test is only performed for 20 rotations, completed in 1 minute (Ozdemir and Nilsen, 1999).

2.2.3.2.6 **Discussion**

The NTH model is based on a vast database of TBM performance in a variety of geological and machine conditions. The model also focuses on whole-system processes, rather than single cutter processes, and their effect on machine performance and can therefore better
take rock mass properties into account. Extrapolation of machine performance into geological conditions that have not been encountered could be unreliable, as with all empirically based models.

2.2.3.3 Colorado School of Mines (CSM) Model

2.2.3.3.1 Introduction

The CSM model is a theoretical model based on the cutting forces acting on the individual cutters, rather than the evaluation of field performance of the machine as a whole system, like the NTH Model (Rostami et al., 1996). This method is basically used to “determine the overall thrust, torque and power requirements of the entire cutterhead” (Rostami et al, 1996). The model was developed using data from linear cutting tests performed over twenty years at the Earth Mechanics laboratory at the CSM.

The rock parameters required to use the CSM model include: UCS, Brazilian tensile strength, density, Cerchar abrasiveness index, punch penetration index and petrographic analysis (Nilsen and Ozdemir, 1993). It is possible to obtain an electronic version of the model into which rock and machine parameters are input and machine performance predictions, such as rate of penetration, are output (Cigla et al., 2001). The input parameters can be varied in order to obtain the optimal machine performance prediction.

2.2.3.3.2 Linear Cutting Tests

Linear cutting tests are performed in a full scale stiff loading frame onto which a disc cutter is mounted and rolled along a rock sample cast in concrete under a specified thrust force, (Cigla et al., 2001). This simulates the cutting process that occurs under a cutter in a TBM cutter head. During the test the forces applied to the cutter are measured by transducer and related to the depth of penetration (kerf depth) of the cutter during the pass.

2.2.3.3.3 CERCHAR abrasiveness index (CAI) value

The CERCHAR abrasiveness index was developed at the CERCHAR institute in France. It consists of scratching a steel needle with a force of 70 N on a piece of rock for a distance of 10 mm (Bamford, 1984). The width of the needle at the end of the test is measured using a
microscope and related to the CAI. The CAI can be related directly to an estimate for the cutting life of the cutter disc.

Plinninger et al. (2003) have shown that the smoothness of the rock sample has a measurable effect on the CAI value obtained for coarse or inhomogeneous rocks. They have suggested that samples should be cut by diamond rock saw and the CAI value obtained corrected by an empirical equation. They also suggest that needles used in testing should be as close to the specified hardness as possible since there are no comparisons of results from tests using different testing needles and results from using different apparatus could be unreliable.

2.2.3.3.4 Punch Test

The punch test is non-standard and involves the pressing of a button indenter into a volume of rock cast in a confining matrix. The displacement of the indenter and the load required to indent are measured with a transducer. A force-penetration curve of the test results is compiled and a best-fit line is drawn through the points. The slope of this line (force/penetration) is used to estimate penetration-thrust values for cutters on a TBM cutter head. This procedure assumes that the penetration index obtained from the punch test is the same as the penetration index obtained from cutter penetration (Dollinger et al., 1998).

2.2.3.3.5 Discussion

The punch test is more useful for low to medium strength rocks than it is for exceptionally high strength rocks because the force-penetration curve for high strength rocks is non-linear. This non-linearity arises from the need to apply a minimum force before increased force will result in significant difference in penetration (Dollinger et al., 1998).

The CSM model is based on linear cutting tests and theoretical force formulations, rather than empirical data, and as such is more capable of adapting to rock conditions that have not previously been encountered. On the other hand, there are many assumptions necessary to relate data obtained from a linear cutting test to the actual performance of a TBM cutter head where the cutters are rolling in a circular fashion and are subjected to imperfect loading due to heterogeneity of the tunnel face.
2.2.3.4  **Q**\textsubscript{TBM} **Model**

2.2.3.4.1  **Procedure**

**Q**\textsubscript{TBM} was developed by Barton (2000) as a comprehensive system for TBM performance prediction based on rock mass classification. The original Q formulation forms the basis of the **Q**\textsubscript{TBM} formulation, to which improvements were made to incorporate factors relevant to TBM tunnel driving. The final **Q**\textsubscript{TBM} formulation requires twenty parameters or dimensions (from Barton, 2000):

- The six original Q parameters RQD, Jn, Jr, Ja, Jw and SRF, with improvements to RQD and (Jr/Ja), where RQDo is taken oriented in the tunnelling direction and (Jr/Ja) can be selected for use in stability or cutter penetration determination
- The four rock strength and rock mass parameters UCS, I\textsubscript{50}, joint inclination $\beta^\circ$ and density $\gamma$, where UCS and I\textsubscript{50} can be taken parallel or perpendicular to fabric where relevant
- The three abrasivity parameters CLI, q and n
- The four dimensions tunnel diameter D, excavation length L, time T and cutter force F.

These parameters are incorporated into either **Q**\textsubscript{TBM} or a gradient m, which takes utilisation into consideration. These are in turn used to estimate penetration rate (m/hr), advance rate (m/hr, taking into account utilisation) and time to advance a certain length of tunnel (hr, taking into account utilisation). This is a fairly new estimation procedure and, as Barton (2000) states, is not complete and is expected to undergo continued refinement.

2.2.3.4.2  **Discussion**

Sapigni et al. (2002) found that when Q and **Q**\textsubscript{TBM} were applied to three tunnels and the predicted machine performance was compared to the actual machine performance, the values predicted by Q had a better correlation to the actual data than did those predicted by **Q**\textsubscript{TBM}. They suggested that based on this result simple indices may give similar or better results than complex ones.
2.2.3.5 Methods using RMR

Sapigni et al. (2002) compared the Rock Mass Rating (RMR) classification of 14km of bored tunnels to the machine performance and found a relationship between the two. They found that machine penetration rate is maximum between RMR 40-70 but decreases at both higher and lower RMR values (Figure 2.2.29). This is reasonable since at low RMR the poor rock mass would lead to instability in the face and the need to reduce thrust to excavate the difficult ground, while at high RMR the massive rock mass would require higher thrust for penetration of the cutter and chip formation through intact rock. This method is not currently state of practice, but may prove useful with further work, although it is not relevant to the thesis work where RMR is >75.

![Figure 2.2.29: Relation between TBM penetration rate and Rock Mass Rating (after Sapigni et al., 2002)](image)

While this relationship is promising, there can be many situations where rocks with similar RMR classifications result in different machine performance. Sapigni et al. (2002) state that modifiers are necessary to properly relate RMR to machine performance. These modifiers would need to take into account rock toughness and non-RMR parameters such as foliation. Both of these modifiers would aid in differentiating rocks that fall in the medium to high RMR range. For example a massive chalk will be easier to excavate than a massive gneiss, and similarly a foliated, massive schist will be easier to excavate than a non-foliated, massive granite.

The identification of the rock properties that lead to differences in machine performance is clearly needed for approaches relating RMR to machine performance, as well as for the methods in current state of practice.
2.2.4 Limitations of State of Practice

Three basic approaches exist in geological engineering for design and research purposes: empirical, experimental and fundamental science based approaches. Empirical approaches are based on large datasets from excavations for which geological conditions, excavation methodology, performance, required support, stand-up time, etc. are available. Charts or equations are then developed using the relationships between the engineering aspects and geological characteristics for use in subsequent projects. This methodology has been widely applied in various aspects of engineering geology design with considerable success. This methodology is rarely used to improve the fundamental knowledge behind the processes described by the charts. For example, stand-up time with respect to excavation dimension is not used to explain the processes leading to stable versus unstable excavations and is rather only used as a reference with which one can predict stability based on a given geometry. The main limitation of empirical methods arises when conditions fall outside of the dataset with which the empirical approach was developed, for example greater depth, unusual stress conditions, rare geological conditions, etc.

Experimental approaches are primarily based on laboratory testing of the engineering aspect of interest, usually with rock samples from the area in which the excavation will be made. The methodology with the most relevance to this research is the linear cutting test developed at the Colorado School of Mines (Nilsen and Ozdemir, 1999; Tapponier and Brace, 1976) to test chipping performance from TBM cutters. This methodology allows the direct testing of the engineering tools to be used in the exact rocks that occur in the field, allowing a direct prediction of anticipated performance. The main limitations to this methodology arise from the inability to completely simulate underground conditions in the laboratory, most importantly in-situ induced stress conditions, as well as humidity, groundwater, temperature, etc. This methodology is also, not typically used to provide a holistic fundamental understanding of the processes involved, mostly due to the lack of simulation of in-situ conditions.

Fundamental science approaches are based on theoretical mathematical formulations used to describe processes related to excavation. For example, Diederichs (2003) used this methodology to explain near excavation boundary instability processes arising from lack of confinement in blocky (Voussoir beam) and massive (spalling limit) rock masses. This was the most recent in a line of rock mechanics research addressing rock fracture processes under compressive and tensile stress conditions, including work by Griffith (1924), Brace (1960), Kuksenko et al. (1996), Lockner and Madden (1991), Tapponier and Brace (1976) and Wawersik
and Brace (1971) to name a few. The main advantage of this methodology is the development of improved theoretical principles as new testing and new empirical data become available. This methodology, by explaining the fundamental processes involved during excavation, is more adaptable to novel conditions than empirical methodologies and provides answers from more situations than can be tested with laboratory methodologies. In order for this methodology to be applicable, however, considerable empirical data are necessary for validation, and laboratory testing is often necessary to observe the processes leading to mathematical solutions.

An engineering geology design methodology based on theoretical formulations, inspired by laboratory testing and validated with empirical datasets is necessary to develop a fundamental understanding of rock behaviour during excavation that can be extended to new conditions as they are encountered. This research includes theoretical formulations for rock excavation by cutters, empirical analysis of TBM performance and geological conditions in four vastly different rock conditions and laboratory testing undertaken in several ways: rock strength testing, testing using a tunnel boring machine and numerical modelling of rock strength testing and excavation using tunnel boring machine cutters.
2.3 Geology and Engineering Geology of Switzerland and NEAT Tunnels

2.3.1 Overview of the Geological Setting and Geological History

2.3.1.1 Introduction

The fieldwork undertaken in this research was conducted in the Central and Southern Alps of Switzerland. The Alps are a 300km long and 100km wide mountain belt located in South-Central Europe from the French and Italian Mediterranean coast to Eastern Austria. A full understanding of the geological setting requires an overview of the geological history of the region, in particular, the orogenies that affected the area. The Alpine orogeny is the most recent orogeny in the Swiss Alps, starting approximately 130 Ma ago, and is responsible for the uplift resulting in the Alpine mountains visible today. The Variscan orogeny, occurring in the Carboniferous, is responsible for some of the basement rocks found in the Alps. This study will consider the Alpine orogeny in detail, and only consider the aspects of the Variscan orogeny as they relate to the rocks involved in the Alpine orogeny.

2.3.1.2 Geological Domains of the Swiss Alps

There are six separate geological domains (Figure 2.3.1), identified by the Mesozoic paleogeography of the units, depending on either the Mesozoic location of basement rocks or depositional location of cover sequences:

1. Foreland
2. Jura
3. Helvetic
4. Penninic
5. Austroalpine
6. South Alpine
7. Apennines

Four of these domains will be considered in detail. The Foreland consists of the rocks in the Schwarzwald (Germany) and Vosges (France), separated by the Rhine graben (Stampfl, 2001). The Jura domain is made of shallow Jurassic carbonate sequences that were deformed in disharmonic folds during the Alpine Orogeny. The Jura domain was the youngest domain to be accreted and deformed, as the deformation
progressed northward from the suture zones in the late Miocene to early Pliocene (Stampfli et al, 2001).

The Helvetic/Ultrahelvetic domain consists of rocks formed on the southern margin of the European continent during the Mesozoic (Trümpy, 1980). The Helvetic domain is mostly made up of carbonate passive margin sequences (Northern portion) while the Ultrahelvetic domain is made up of more distal carbonate sequences overlaid by pelagic sediments (Southern portion). The Helvetic/Ultrahelvetic massifs formed by magmatism during the subduction of the Paleotethys ocean in the Permian (Stampfli, 2001). The Helvetic/Ultrahelvetic Nappes formed by thin skin deformation during underthrusting of the basement during the Alpine orogeny. The Ultrahelvetic Nappes are commonly found on top of the Helvetic Nappes and in some cases are wrapped around them.

The Penninic domain consists of the very distal part of the European margin, oceanic (Piemont ocean) and para oceanic furrows (Valais trough) and intervening platforms and swells (Brianconnais) that formed in the Jurassic (Trümpy, 1980). The Penninic domain was deformed into Nappes after being detached from the basement by the advancing Austroalpine domain in the Eocene (Hsü, 1995). Highly metamorphosed (in Tertiary) Penninic basement was uplifted in the Miocene (Hsü, 1989).

The Austroalpine domain consists of rocks from the southern passive margin and continental shelf of the Adriatic plate formed in the Jurassic as well as early Paleozoic volcanics.
The Austroalpine domain was back folded as an accretionary wedge during the northern movement of the Adriatic Plate in the late Cretaceous (Stampfli, 2001). It was then overthrust onto the Helvetic/Ultrahelvetic in the Eocene and was, therefore, less metamorphosed than the Penninic domain (Hsü, 1995).

The South Alpine domain consists of rocks that formed the northern margin of the Adriatic plate (separated from Northern Africa). It includes the southern part of the Austroalpine domain but deformation there is more simple and only consists of south facing faults and folds (Trümpy, 1980).

### 2.3.1.3 Geological History of the Swiss Alps

The Geological history can be summarised as follows:

1. Variscan orogeny – docking of the Gothic terrane
2. Rifting and collapse of the Variscan orogen
3. Subduction and closure of the Paleotethys, formation of Pangea and opening of back arc basins
4. Opening of the Alpine Tethys between Europe and Africa
5. Separation of Italy and the Balkans from Africa as the Adriatic plate
6. Accretion of exotic terrains onto the Adriatic plate during northward movement
7. Collision between the Adriatic plate and Europe
8. Deformation of the Alpine Nappe sequences
9. Quaternary glaciation, erosion and uplift

The opening of Paleo-Tethys in the Devonian (ca. 400Ma) led to the accretion of the Gothic terranes from Gondwana to the Laurasian plate and the creation of the Variscan Cordillera in the Carboniferous (390-310Ma) (Trümpy, 1980). The closure of the Paleo-Tethys led to magmatism on the European margin and the collapse of the Variscan Cordillera and transgression of the former Variscan relief in the Permian (ca 270Ma). Many of the Alpine massifs were created during this period of magmatism (Schaltegger and Gebauer, 1999; Stampfli, 2001; von Raumer et al, 1999).

In the Early Triassic to Mid Jurassic (250-170Ma) the deposition of cover sequences of carbonate platform type took place (Trümpy, 1980). Italy and Balkans broke off from Africa to form the Adriatic ‘micro’ plate in the late Triassic. The opening of the Alpine Tethys ocean in the Jurassic is associated with opening of Central Atlantic Ocean and the breakup of Pangea (Stampfli, 2001). From the Mid Jurassic to mid Cretaceous (170-100Ma) the platforms separated into the Helvetic (northern margin), Penninic (central basins and platforms) and
Austroalpine/Southern Alps (southern margin), followed by a transgression and the deposition of the Jura and Molasse basins (Trümpy, 1980).

The Adriatic plate began its northward movement ahead of the African plate in the late Jurassic to early Cretaceous (ca. 150Ma). Subduction of the Alpine Tethys due to northward movement of Adriatic plate occurred in the mid Cretaceous (ca. 100Ma) (Stampfli, 2001). Uplift of ophiolite sequences occurred during the northward movement of the Adriatic plate and the closure of the Alpine Tethys Ocean (Hsue, 1995). Back folding of Austroalpine orogenic wedge occurred in the late Cretaceous (ca. 70Ma) (Stampfli, 2001).


Deformation and subsidence of the Molasse into the Molasse basin and the beginning of sedimentation of Alpine clastics into the Molasse basin occurred in the early Miocene (ca. 23Ma) (Stampfli 2001; Hsue, 1995). Jura accretion and deformation occurred during the late Miocene to early Pliocene (7-3.5Ma) after detachment from and underthrusting of the European basement. Deformation and uplift of the Aar Massif (Figure 2.3.2) due to the warping of the underlying, advancing crustal wedge occurred in the Pliocene (ca. 5Ma) (Gebauer, 1999; Schmid, 2004; Trümpy, 1980).

The morphology of the Alps was largely shaped by Pleistocene glaciers. The Molasse Basin is filled with moraines and glacial deposits. Two episodes of glaciation produced high plateau gravels while an interval of intense erosion carved the present river valleys and lake basins. The Riss glaciation (1.3 Ma) deposited moraines and gravel terraces in river valleys and over-deepened depressions. Only slight erosion occurred in the interglacial. The Wurm glaciation (70 ka) deposited moraines and sandur gravels and the present rivers have only cut into Wurm deposits by a few metres. Many mountains collapsed after the glacial retreat due to the removal
of supporting pressure from glaciers (17 ka), leaving slide debris in many of the Alpine valleys (Trümpy, 1980). Current uplift of the Alps continues, on the order of 1.6mm/year in some areas, due to continued uplift and deformation of the Aar Massif and movement along post-glacial faults (Persaud and Pfiffner, 2004).

2.3.1.4 Geology of the Central and Southern Swiss Alps

2.3.1.4.1 Central Swiss Alps

The central Swiss Alps encompass the area between Thun (west) and the Reuss valley (east) and just north of the Gotthard Massif and Valais trough (Rhone river). This region is dominated by the Aar Massif and the Helvetic nappes, and also the Ultrahelvetic and Prealpine nappes in small outliers.

The Aar Massif (autochthonous) is composed of various gneisses in the north, the central Aar granite to the south and various gneisses and granitoids on the southern margin. It has undergone shortening and thickening but was not affected by intense tectonic metamorphism. Its sedimentary cover has been almost completely removed (Hsue, 1995).

North of the Aar Massif are the Gastern and Erstfeld Massifs, made up of gneisses and Mesozoic sediments, and deformed by north facing folds. North of these massifs is the Wildhorn nappe, made up of Jurassic Cretaceous Helvetic cover rocks (Masson, 1980). West of the Aar Massif is the Doldenhorn nappe, made up of the folded cover rocks of the Aar massif. Ultrahelvetics are only sparsely represented. Parautochthonous slices occur between the Aar Massif and Helvetic Nappes (Trümpy, 1980).
2.3.1.4.2 Southern Swiss Alps

The southern Swiss Alps encompass the rocks of the Ticino Canton, south of the Aar Massif. This region is dominated by the Tavetsch and Gotthard Massifs, some Mesozoic rocks and the large gneiss units of the Lepontine (Penninic domain). This region also contains the Periadriatic faults separating the Penninic domain from the South Alpine domain.

The Tavetsch Massif represents the remainder of the basement of the Helvetic domain (originally ~35km wide N – S). The Helvetics were detached and much of the Tavetsch massif and the rest of the Helvetic basement (originally ~100 km wide N S) was subducted under the Gotthard Massif in continental subduction (Trümpy, 1980 and Hsue, 1995).

The Gotthard Massif (parautochthonous) is a Variscan granitoid that was thrust onto the Tavetsch, which was in turn thrust onto the Aar Massif. Its cover rocks are of Ultrahelvetic domain, which have for the most part been removed and those that remain are very deformed (Hsue, 1995). The Massif was moderately metamorphosed and its schistosity has a fan like internal structure opening upwards (Trümpy, 1980).

The Lepontine dome consists of a folded series of Variscan basement gneisses and Mesozoic sediments. It can be divided into two subdomes: Simplon and Ticino, separated by the Maggia steep zone. The axis of the Ticino dome is the Leventina valley between Airolo and Bellinzona. The northern steep zone abuts the Gotthard Massif while the southern steep zone corresponds to the Periadriatic faults (Merle, Cobbold and Schmid, 1989).

The Periadriatic faults are post metamorphic and occurred late in the Alpine orogeny (Merle et. al., 1989). Metamorphism increases southward from greenschist at the northern part of the Gotthard Massif to amphibolite grade along most of the Ticino Alps. The highest grade of metamorphism is found just north of the Periadriatic faults (Trümpy, 1980). Metamorphism in this area corresponds to subduction of the basement rocks during the Alpine orogeny.

2.3.1.5 Geology of the Gotthard Transect

The majority of the research was conducted in the Gotthard tunnel, with some work conducted in the Lötschberg tunnel. Due to the importance of the data collected in the Gotthard tunnel, a more in-depth description of the units encountered is included here. The Gotthard tunnel transects the Central and Southern Swiss Alps from North to South. The geological units encountered in the Gotthard tunnel are, from North to south (Figures 2.3.3 and 2.3.4):
1. Aar Massif
2. Disentis syncline/ Clavaniev zone
3. Tavetsch Massif
4. Unseren/ Garvera syncline
5. Gotthard Massif
6. Piora synform
7. Lucomagno unit
8. Leventina unit

Figure 2.3.3: Geological map of Gotthard pass with the Reuss valley to the North and the Ticino valley to the South (after Schweizerischen Geologischen Kommission, 1980a and 1980b).
Along the Gotthard transect the Aar Massif is comprised of northern and southern gneiss zones and a core of Central Aar Granite, all of Variscan granite origin. The external zones underwent the greatest amount of metamorphism while the core underwent very little metamorphism. Alpine foliation generally dips to the south, becoming steeper and more intense southwards. Alpine metamorphism generally increases southwards. The northern margin contains gneisses that belong to a frontal overfold of the massif (Trümpy, 1980). The units of the Aar Massif transected by the Gotthard tunnel (from N to S) include (Figures 2.3.5 and 2.3.6): Erstfelder gneiss, Altkristallin, the Intschi zone, Altkristallin, the Tscharren group, the Central Aar granite, the Northern Schollenzone and the Giuv syenite, the Southern Aar granite, the Southern granitic gneiss and the Southern gneiss zone (Keller, 1999). The Central Aar granite was the only unit encountered during the fieldwork in the Lötschberg tunnel, and is genetically related to the Southern Aar granite.

Figure 2.3.4: Cross section through the Gotthard pass (after Keller, 1999)

Figure 2.3.5: Detail from Figure 2.3.2 showing the different rock units within the Aar Massif
The Disentis syncline is composed of steeply dipping cataclastic fault rocks of the Clavaniev zone, the Bugnei granodioritic and gneissic core and the Mesozoic rocks of the Disentis zone. The tunnel itself is just west of the extension of the Bugnei zone and will not likely encounter it (Keller, 1999). These rocks abut the basement rocks of both the Aar and Tavetsch Massifs (Trümpy, 1980). Topographically, they correspond to the Oberalp pass. The Mesozoic sequence contains (from N to S) a Dogger Malm sequence enclosed by two Triassic sequences, a Permo-Carboniferous sequence and a Ceratophyre sequence (Hsue, 1995).

The Tavetsch Massif is believed to be Helvetic basement rocks (Trümpy, 1980). These were underthrust and almost completely subducted below the Gotthard Massif, before being overthrust onto the Aar Massif. As a result these rocks are very highly deformed and sheared. They are composed of (from N to S) phyllonite with serpentine, paragneiss, amphibolite and diorite (Hsü, 1995).

The Unseren Garvera syncline is composed of steeply dipping late Paleozoic, Triassic and early Jurassic formations. These rocks represent the overturned Ultrahelvetic cover rocks from the Gotthard Massif (Trümpy, 1980) and their location corresponds to the location of the Sedrun shaft. This unit contains (from N to S) a phyllite, Liassic sequence, Triassic sequence, Permo Carboniferous sequence, quartz porphyry and a schist (Hsue, 1995).

The Gotthard Massif is composed of Variscan age basement granites that have been more or less affected by Alpine metamorphism to greenschist facies. They form part of basement of the Ultrahelvetic domain which was overthrust onto the Tavetsch Massif, and in turn the Aar Massif. The massif contains (from N to S) paragneiss (sedimentary), orthogneiss (igneous),
granite and granodiorite and granite (Hsue, 1995). To the east of the Gotthard transect (Lucomagno pass) the Ultrahelvetic cover sequences are visible and form the Scopi synform and Piora synform (to the south). The schistosity planes are steeply inclined in a fan like structure that opens upwards (Trümpy, 1980).

The Piora synform is composed of Triassic and lower Jurassic sequences that enclose Pennine Buendnerschiefer and it separates the Gotthard Massif from the Penninic basement rocks of the Lucomagno unit. The Triassic sequences are in contact with basement rocks but the Jurassic sequences are inverted. The detachment took place before the deformation, likely concurrently with the arrival of the Buendnerschiefer sheets. Metamorphism and deformation were likely simultaneous and may be linked with the underthrusting of the Tavetsch beneath the Infrapenninic basement (Trümpy, 1980).

The Lucomagno unit is a lower Penninic granitic gneiss of the Lepontine dome, located above the Infrapenninic Leventina gneisses. Its cover was Ultrahelvetic but the deformation was Penninic (Lepontinic) (Trümpy, 1980). The boundary between the Lucomagno and the Gotthard is a region of backfolding, where the backfolds (Figure 2.3.7) are late structures post-dating the high grade metamorphism (Merle et. al., 1989).

The Leventina unit is also a lower Penninic granitic gneiss of the Lepontine dome (Figure 2.3.8), located below the Infrapenninic Lucomagno gneisses and forms the base of the Leventina valley and the hinge of the Lepontine dome (Merle et. al., 1989). Its cover units are unknown. It was overridden by the large, recumbent Simano nappe (Trümpy, 1980), which forms a gently dipping anticline.

Figure 2.3.7: Schematic cross-section of the northern end of the Lucomagno gneiss showing the steep backfolding. Large arrows are shear indicators from earlier high temperature and pressure event, while small arrows denote late retrograde shearing associated with the backfolding (after Merle et. al., 1989)
2.3.2 Engineering Geological Description of the Project Rocks

Research was conducted in four geological units in the Gotthard and Lötschberg tunnels: Altkristallin, Southern Aar granite, Leventina gneiss (Gotthard, Figure 2.3.9) and Central Aar granite (Lötschberg) (see Appendices 2.3 and 2.4). Each unit has a different geological provenance and history, leading to difference in micro and macro geological characteristics, which are described in the following sections. In addition to the alpine metamorphism, the alpine tectonic overprint in the north part down to the southern border of the Aar-Massif was very weak (epimetamorphic, greenschist facies). In the area of Amsteg some of the rocks underwent extensive pre-alpine deformation from which the structure and minerals remain. The alpine metamorphism in the part north of the Maderenertal is predominantly retrograde, and it caused a green colour in many places due to chlorite alteration. South of Amsteg all of the alpine deformed rocks have been in part recrystallised. The alpine overprint is selective and is manifested as single massive parallel zones that received stronger metamorphism, such as the Intschi-zone and the southern part of the Aar-Massif (Keller, 1999).
2.3.2.1 Micro and Macro Structure and Texture of the Project Rocks

2.3.2.1.1 Altkristallin

The Altkristallin unit is part of the Aar Massif, which is a collection of intrusive and metasedimentary rocks of varying ages that were thrust 25-50km during the advancement of the Alpine orogeny (Burkhard and Isler, 2005). The Altkristallin are complex pre-Variscan metasedimentary basement rocks that have undergone contact (intrusion of Aar granite) and regional (Caledonian, Variscan, Alpine) metamorphic episodes and consist of highly foliated to migmatitic, granitic (metatectic migmatites) to tonalitic (diatectic migmatites) gneisses (Abrecht, 1994; Schaltegger, 1993; Schaltegger, 1994). The foliation varies slightly from schistosity to cleavage, with microlithon spacing between 0.5 to 5mm (Figures 2.3.10 and 2.3.11).

The Altkristallin is made up almost exclusively of quartz, biotite, k-feldspar and plagioclase (Figure 2.3.11). The majority of the feldspar grains, in particular the plagioclase, have been altered to some degree to sericite (Figure 2.3.12) arising from strong Variscan and Alpine retrogression and deformation (Abrecht, 1994; Schaltegger, 1993; Schaltegger, 1994). The foliation varies slightly from schistosity to cleavage, with microlithon spacing between 0.5 to 5mm (Figures 2.3.10 and 2.3.11).

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structure in the black material is shown in Figure 2.3.13, composed of finely ground minerals in a folded band. Some samples contained fresh fractures arising either from the excavation or sample collection process (Figure 2.3.13).

Figure 2.3.10: Typical photos of the excavated surface of the Altkristallin.
Figure 2.3.11: Typical photomicrographs of Altkristallin (GA-031, GA-037, GA-013, GA-102)
Figure 2.3.12: Photomicrographs of top left: sericite alteration of plagioclase (GA-049); top right: subgrain formation in quartz grains (GA-095); bottom left: quartz ribbon with black material along top of grain (GA-013); bottom right: mica grains and black material defining fabric (GA-095).

Figure 2.3.13: Photomicrograph of left: dark dissolved material hosting folded band of crushed relic grains (GA-095); right: macro fracture through sample (GA-049).

2.3.2.1.2 Southern Aar Granite

The Southern Aar granite (SAG) is a part of the Variscan calc-alkaline to subalkaline granitic intrusives made up of granites, granodiorites and leucogranites dated to 298Ma (Schaltegger, 1993; Schaltegger, 1994). These intrusives postdate the Variscan deformation, but were affected by the Alpine deformation. The SAG encountered during the research is mostly composed of granitic gneiss with quartz, feldspar and low to moderate (5-25%) mica content (Keller 1999) making up the major mineralogical components. The average quartz content is 30%, with feldspar content around 50%. Some more granitic sections contain closer to 50%
quartz, nearly 50% feldspar and less than 5% mica. More micaceous sections contain less than 15% quartz, nearly 50% feldspar and over 40% mica. The fabric varies from granitoid to intensely foliated schist on a scale from one metre to several tens of metres (Figure 2.3.14). The research was conducted near the southern boundary of the SAG and the Southern granite gneiss (SGG), although the boundary is not clear and the only distinction is an increase in deformation and metamorphic alteration (Abrecht, 1994). It is not certain whether the material collected was from the SAG or SGG, although it has been suggested that the SGG is simply more deformed SAG (Abrecht, 1994), rendering the distinction unnecessary.

The grain size distribution is as variable as the fabric from bimodal to isotropic, with an average grain size of 1mm. As seen in Figure 2.3.15, the feldspars can form porphyroclasts, either with considerable rounding and aligned with foliation or angular with random orientation. Some samples also have wings composed of quartz, mica and altered feldspar (Figure 2.3.16). The quartz grains are highly deformed and show signs of plastic deformation, including subgrain boundary and quartz ribbon formation (Figure 2.3.16), as seen in other Alpine massifs (Voll, 1976; von Raumer, 1984). The mica grains are typically less than 0.5mm, can be euhedral or disseminated and often define the fabric (Figure 2.3.16). In highly cleaved material the mineralogy is nearly indiscernible due to the extreme small grain size. Silimanite (Vinall, 2006) and calcite (Figure 2.3.17) were often present, suggesting that some portions of this rock unit are metasedimentary. Many samples contained fresh fractures arising either from the excavation, sample collection process or prior strength testing (Figure 2.3.16).
Figure 2.3.14: Typical photos of the excavated surface of the Southern Aar granite.
Figure 2.3.15: Photomicrographs of texture and mineralogy corresponding to excavation boundary photos in Figure 2.3.14 (GA-b034, GA-a071, GA-a066, GA-a068).
Figure 2.3.16: Photomicrographs of top left: feldspar porphyroclast with quartz-mica-feldspar wing (GA-b095); top right: highly deformed quartz ribbon (GA-b095); bottom left: mica grains defining fabric (GA-a121); bottom right: fresh fracture through mica and around feldspar grain (GA-a093).

Figure 2.3.17: Photomicrographs of left: very fine, nearly indiscernible mineralogy in highly cleaved material (GA-a102); right: calcite grains denoted by “C” (GA-a091)
2.3.2.1.3 Leventina Gneiss

The Leventina unit, in locations where it is not affected by brittle deformation features, is hard, massive granodioritic gneiss (Figures 2.3.18 and 2.3.19) that has been metamorphosed to amphibolite facies (Zappone, Sciesa and Rutter, 1996). The basement of the Penninic domain, to which the Leventina unit belongs, was subducted during the advance of the Alpine orogeny and subsequently uplifted, perhaps by denudation, as the orogeny progressed (Beaumont et al, 1996). This may account for the amphibolite facies deformation experienced by the Leventina unit, suggesting high temperature and pressure deformation. The portions of Leventina gneiss that do not exhibit macro-scale brittle deformation features (i.e. large shears and faults) are much stronger, and are therefore of interest to this research. The mineral composition is typically 35% quartz, 55% feldspar and less than 10% mica.

Figure 2.3.18: Typical photos of the excavated surface of the Leventina gneiss.
The grain size distribution is isotropic and seriate, with an average grain size of 0.8mm, although in some cases the distribution is bimodal with feldspar porphyroclasts approaching 10mm (Figure 2.3.20) aligned with foliation. Considerable deformation of the feldspars is visible in the presence of mechanical twins, which are in some cases bent (Figure 2.3.20). The quartz grains are highly deformed and show signs of plastic deformation, including subgrain boundary and quartz ribbon formation (Figure 2.3.20). The mica grains are typically less than 0.5mm, can be euhedral or disseminated and often define the fabric (Figure 2.3.21). Many samples contained fresh fractures arising either from the excavation, sample collection process or prior strength testing (Figure 2.3.21).

Figure 2.3.19: Typical photomicrographs of Leventina gneiss (GB-01, GB-03, GB-05, GB-08).
Figure 2.3.20: Photomicrographs of top left: feldspar porphyroclast (GB-01); top right: quartz ribbons (GB-02); bottom left: recrystallised quartz (GB-07); bottom right: bent mechanical twins in feldspar (GB-09)

Figure 2.3.21: Photomicrographs of left: mica grain texture (GB-11); right: fresh fracture through sample (GB-11)
2.3.2.1.4 Central Aar Granite

The Central Aar granite (CAG) is likely contemporaneous to the SAG (Abrecht, 1994) and is a complex of at least nine identified granitic bodies made up of coarse to fine-grained, granodiorites to leucocratic granites, from massive to strongly foliated (Debon and Lemmet, 1999). The samples collected during this research are massive and contain no discernable foliation (Figures 2.3.22 and 2.3.23). The mineral composition is typically 30% quartz, 60% feldspar and less than 5% mica.

The grain size distribution varies from seriate to bimodal, with an average grain size of 1.5mm. As seen in Figure 2.3.23, the feldspars are angular, with ragged boundaries and often contain microfractures. The quartz grains are deformed and show signs of plastic deformation, including subgrain boundary and chessboard texture (Figures 2.3.23 and 2.3.24). The mica grains are typically less than 0.5mm and tend to be disseminated (Figure 2.3.23). Many samples contained fresh fractures arising either from the excavation, sample collection process or prior strength testing (Figure 2.3.24).

Figure 2.3.22: Typical photos of the excavated surface of the Central Aar granite.
2.3.2.2 Excavation Boundary Instability Features

The research locations were selected for their massive and stable characteristics in order to conduct research regarding the impact of intact rock properties on TBM excavation. Figures
2.3.25 to 2.3.28 demonstrate the typical excavation boundary characteristics in each of the rock units introduced in Section 2.3.2.1. Some locations were more adversely affected by instability issues, but these were uncommon and of limited extent. The instabilities arose from two separate processes: gravity falls of blocks defined by pre-existing joints and fractures (Figure 2.3.29) and stress-related instability due to spalling or strain bursting (Figure 2.3.30) in particular in the Leventina gneiss and Central Aar granite.

Figure 2.3.25: Typical excavation boundary in Altkristallin.

Figure 2.3.26: Typical excavation boundary in Southern Aar granite.
Figure 2.3.27: Typical excavation boundary in Leventina gneiss.

Figure 2.3.28: Typical excavation boundary in Central Aar granite after one year (left) and two years (right) of exposure.

Figure 2.3.29: Gravity rock block fall at the excavation boundary arising from blocks defined by joints and fractures in Southern Aar granite.
2.3.2.3 Domain Classification in the Southern Aar Granite

A 400m length of the Southern Aar granite was more intensely sampled than elsewhere, including samples at 1m intervals that were point load tested and for thin sections at 2m intervals (see Appendix B.3). In order to relate the point load strength tests to sparse UCS tests, as well as investigate meso-scale geological impact on TBM excavation, the dataset was divided into 10 domains (“A” to “J”) with similar geological texture, mineralogy, fabric, fractures and excavation boundary characteristics as well as the frequency of variability of each of these characteristics within a domain (Table 2.3.1). These domains are shown with their associated characteristics in Appendix B.4.
Table 2.3.1: Description of geological domains.

<table>
<thead>
<tr>
<th>Domain</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>&gt;10 metre scale variability with feldspar, quartz and mica, in decreasing percentage, medium (0.5-5mm) grain size and fabric ranging from preferred orientation of feldspars to schistosity defined by micas, no spalling</td>
</tr>
<tr>
<td>B</td>
<td>&gt;10 metre scale variability with feldspar, quartz and mica, in decreasing percentage, medium (0.5-5mm) grain size and fabric consisting of schistosity defined by micas, approximately 5% spalling</td>
</tr>
<tr>
<td>C</td>
<td>&gt;10 metre scale variability with feldspar, quartz and mica (up to 30%), in decreasing percentage, small (&lt;0.5mm) grain size and fabric consisting of continuous cleavage defined by micas, nearly 30% of area contains spalling</td>
</tr>
<tr>
<td>D</td>
<td>Decametre scale variability with feldspar, quartz and mica (greater than 10%), in decreasing percentage, medium (0.5-5mm) grain size, with large feldspars (10-20mm), infrequent joints, 5-10% of mapped area contains spalling, varying fabric from none to domainal schistosity</td>
</tr>
<tr>
<td>E</td>
<td>Less than decametre scale variability with feldspar, quartz and mica (high variability from 2-25%), in decreasing percentage, medium (0.5-5mm) grain size, with micas often &lt;0.5mm and feldspars often &gt;5mm, and fabric ranging from preferred orientation of feldspars to schistosity defined by micas to cleavage defined by micas in narrow shear zones, 10-25% of mapped area contains spalling</td>
</tr>
<tr>
<td>F</td>
<td>Decametre scale variability with feldspar, quartz and chlorite (5-7%), in decreasing percentage, large (&gt;5mm) grain size, no fabric, jointing or shear zones, discing of the core</td>
</tr>
<tr>
<td>G</td>
<td>&gt;10 metre scale variability with feldspar and quartz (mica only ~2%), medium (0.5-5mm) grain size and no fabric, no spalling and infrequent shear zones</td>
</tr>
<tr>
<td>H</td>
<td>Decametre scale variability with equal feldspar and quartz, and slightly less mica (25-30%), medium grain size (0.5-5mm), fabric varying from closely spaced cleavage to intermediate spaced schistosity, no fractures, shears with 5-10m spacing, some core discing</td>
</tr>
<tr>
<td>I</td>
<td>&gt;10m scale variability with feldspar, mica and quartz in decreasing percentage, small grain size (&lt;0.5mm), closely spaced cleavage, no fractures or shears, 5-10% of tunnel wall with spalling</td>
</tr>
<tr>
<td>J</td>
<td>Decametre scale with feldspar, quartz and mica in nearly equal percentage, small grain size (&lt;0.5mm), closely spaced cleavage, shear zones with &gt;10m spacing</td>
</tr>
</tbody>
</table>

2.3.3 The Incompatibility between Geological Science and Engineering Geology in the Current State of Practice

A review of recent deep TBM tunnelling projects in the Swiss Alps has shown that, except for in a few extreme cases (Bonzanigo and Opizzi, 2005; Burkhard and Isler, 2005), the geological prediction was similar to the rock actually encountered during tunnelling (Chopin, 2005; Frei and Breitenmoser, 2005). The selection of appropriate excavation and support tools and techniques critically depends on adequate prediction of rock mass behavior in response to tunnelling for each geological domain within the geometrical and mechanical tunnelling framework (Kaiser, 2005). The need for quantifying geological descriptions for engineering geology applications such as open pit mine wall stability (Hoek, 1999) and deep, hard rock, tunnel stability (Kaiser, 2005) has been demonstrated but to effectively accomplish this, the rock behaviour and response must first be understood in order to define the values of importance for quantification (Kaiser, 2005). In-situ behaviour can vastly differ from laboratory behaviour, depending heavily on textural properties (Diederichs et al., 2004), making understanding rock behaviour at the excavation boundary critical to properly quantifying geological characteristics.
TBM design and performance prediction requires appropriate predictions for rock behaviour and response based on the geological data available. The current state of practice does not provide a framework in which the geological information contained in tender documents can be translated for application. Conversely, due to the lack of a framework, the appropriate geological information is not necessarily collected, interpreted and presented for design. A methodology by which the critical geomechanical characteristics are identified and related in a framework leading to rock behaviour prediction is needed. This research includes the construction of a geomechanical characterisation scheme to translate information available through geological description into information that relates directly to rock behaviour, focusing on its impacts on chipping and face instability.
2.4 Geomechanical Characterisation Methodology for Deep Tunnelling

2.4.1 Introduction

In order to improve the ability for field geological data to be used to predict rock behaviour during excavation a geomechanical characterisation scheme was developed based on identification of the critical geological parameters for rock yielding. These parameters were identified using published investigations of analysis of the grain scale geological characteristic dependence of fracture behaviour and parametric analysis of geological characteristic dependence of laboratory strength values. The critical geological parameters that were identified were then investigated for their impact on numerical simulation of laboratory strength tests and numerical simulation of the chipping process and stress-related preconditioning. These parameters were also investigated for their impact on chipping performance and face stability of field TBM performance data from the Southern Aar granite.

2.4.2 Chipping Performance

The critical parameters, mineralogy, grain size and grain size distribution, and fabric type and intensity, through parametric analysis of their impact on the chipping process, were divided by thresholds for behaviour and assigned F-Factor (FM, FG and FA, respectively) values weighted according to impact on the chipping process. These factors were assigned values between 0.7 and 2 (where 1 is neutral impact), and were multiplied such that each factor progressively increased or decreased the product, termed the Spalling Sensitivity, \( F_{SSA} \). By this methodology, each parameter category has an equal impact on the Spalling Sensitivity with the values reflecting the magnitude of impact for each parameter separately. If the values were allowed to approach zero, then one factor could hypothetically dominate over the other factors. The in-situ stress condition was tested using numerical simulation of a stressed rock block with oriented fabric and associated with a Stress-Related Chip Potential Factor, \( S_{CP} \).

The rock strength (measured using UCS, Brazilian tensile strength and point load index strength) is also critical to chipping performance, but cannot be adequately related to chipping performance, as discussed in Section 2.3. The Spalling Sensitivity is used to modify the strength
value to improve chipping performance prediction. The Spalling Sensitivity, as a value distributed around 1 is used to reduce or increase the strength value through multiplication. This

Figure 2.4.1: Flowchart demonstrating input data for calculating Chipping Resistance Factor $C_R$.

Table 2.4.1: Summary of thresholds for Chipping Resistance Factors.

<table>
<thead>
<tr>
<th>Chipping Resistance Factor Type</th>
<th>Low Risk of Poor Chipping</th>
<th>Moderate Risk of Poor Chipping</th>
<th>High Risk of Poor Chipping</th>
</tr>
</thead>
<tbody>
<tr>
<td>$F_{SSA} = F_A \times F_G \times F_M \times S_{cp}$</td>
<td>&lt;1.5</td>
<td>1500-4000</td>
<td>&gt;4000</td>
</tr>
<tr>
<td>$C_R = F_{SSA} \times UCS \times BTS$</td>
<td>&lt;1500</td>
<td>1500-4000</td>
<td>&gt;4000</td>
</tr>
<tr>
<td>$C_R = 750 \times F_{SSA} \times PLT_d$</td>
<td>&lt;1500</td>
<td>1500-4000</td>
<td>&gt;4000</td>
</tr>
</tbody>
</table>

resulting value is termed the Chipping Resistance Factor, $C_R$, (Figure 2.4.1), which was related to chipping performance using numerical simulation of the chipping process and field data. Thresholds have been identified for the Chipping Resistance Factor, using both UCS and Brazilian tensile strength or point load index strength, at which poor chipping performance becomes more likely (Table 2.4.1).

2.4.3 Face Stability
Numerical simulation of rock blocks subjected to varying stress conditions and fabric orientations were used to determine sensitivity to stress preconditioning of the rock, which may promote face instability. The magnitude of preconditioning was analysed qualitatively and used to identify fabric orientation and stress condition combinations that result in more severe preconditioning, and thus high potential for face instability. Fabric types were also investigated using the same methodology. Look-up tables were created by which higher potential for TBM performance reduction arising from face instability could be identified based on fabric type, orientation and in-situ stress condition.

2.4.4 Field Data Collection

By identifying the geological parameters critical to chipping performance, it should be possible to use the considerable volumes of geological information available in tender documents to identify potential zones of low chipping performance or face instability. It also makes it possible to identify the geological data that should be requested for inclusion in the tender documents. Two data collection methodologies are proposed, depending on the availability of data: intensive sample collection and thin section analysis or domain classification. If sample collection at the TBM stroke scale is possible, then samples should be thin sectioned and the geomechanical parameters should be collected for Chipping Resistance Factor calculation and estimation of face instability potential. If data is sparse, or only field mapping is possible, then data should be grouped into domains with similar characteristics and variability; a range of geomechanical factors should be estimated for each domain and used to determine a range of Chipping Resistance Factor and face instability potential.

Alternatively, the domain classification can be used to identify potentially problematic zones, in terms of chipping performance, which could then be further sampled for a more detailed investigation using thin section F Factor data collection. In zones of highly variable stress condition or fabric orientation, the same process of domain classification and identification of potentially problematic zones could be undertaken to perform more precise stress measurement or estimation and fabric orientation measurement.

Simple tables were devised specifying thresholds and associated F-Factor values that should be associated with each geological parameter. Once the geological data, from thin sections or domain ranges, has been collected, then F-Factors can be looked-up in the charts and multiplied to obtain the Spalling Sensitivity. The Chipping Resistance Factor is then calculated using laboratory strength values, fabric orientation and in-situ stress condition, if available. If
they are not available, the Spalling Sensitivity can also be used to estimate chipping performance, but is less precise.

The tables devised for face instability cannot at this point be combined for a quantitative estimate of reduced TBM performance, however they can be used to identify conditions under which risk of reduced TBM performance is elevated. These locations can be further investigated by precise stress condition testing or estimation and fabric characterisation, which can then be analysed by detailed numerical simulation.
Chapter 3: TBM Excavation of the Swiss Alpine Tunnels

TBM data collection and analysis were undertaken in the Swiss Alpine tunnels with strong and enthusiastic support from Herrenknecht and on-site contractor consortiums. The data collection methodology was developed, refined and managed on-site. The data analysis was undertaken during research phases in Canada and was heavily influenced by the experiences gained by being physically present during data collection. The data analysis led to an improved understanding of chipping processes and the dependence of chipping performance on geological conditions at the tunnel face.

3.1 TBM Data Collection in the Alpine Base Tunnels

3.1.1 TBM Performance and Geological Data Collection

In order to develop a full understanding of TBM performance in the Alpine tunnels field work was conducted at the Amsteg and Bodio worksites of the Gotthard tunnel and the Raron/Steg worksite of the Lötschberg tunnel. The field work involved data collection from the TBM as well as the exposed rock inside the excavated tunnel (Figure 3.1.1 left). In addition, considerable time was spent on the TBM in various locations in order to observe and experience the excavation process, including:

- the operator’s cabin (Figure 3.1.1 right) to observe the machine operation and inquire regarding TBM operation in different ground types
- the open rock area behind the shield and near the grippers (Figure 3.1.2) to record the wall conditions and perform geological mapping
- at the conveyor belt (Figure 3.1.3) to examine the nature of the excavated material and remove samples
- inside the TBM head (Figure 3.1.4) to examine the tunnel face condition (left), the status of the cutters (right)
- through the TBM head (Figure 3.1.5) to examine the status of the cutterhead from the front (Figure 3.1.6)
- examination of the entire tunnel face (Figure 3.1.7 top) during TBM cutterhead revision (Figure 3.1.7 bottom)
• observation of the excavation process from in front of the TBM (Figure 3.1.8) during excavation of the invert remaining from a rescue operation to release a TBM
• operation of the TBM (Figure 3.1.9) during the performance of start-up tests specially developed for this research.

The understanding of the excavation process acquired during field work was critical in identifying the TBM performance parameters and geological characteristics that are most significantly related to the performance difficulties encountered during Alpine tunnelling. This is the foundation for the laboratory, computer and modelling work undertaken in this research. The majority of this onsite work was undertaken at the Amsteg worksite, while none of this was undertaken at the Raron/Steg worksites because the geological section investigated had previously been excavated and only geological mapping and sample collection could be done.

Figure 3.1.1: Left: inside the exposed rock portion of the TBM (at the invert); right: experiencing the excavation process in the operator cabin.

Figure 3.1.2: Left: inside the exposed rock portion behind the TBM head (background); right: observing the gripping procedure.
Figure 3.1.3: Photographs of large chips (left) and chips and fines (right) on the conveyor belt.

Figure 3.1.4: Left: Examining the tunnel face condition; right: examining the status of the cutters.

Figure 3.1.5: Climbing through the TBM cutterhead (left) to pass through the cutterhead (right)
Figure 3.1.6: Examine of the TBM cutterhead from the front (bottom).

Figure 3.1.7: Top: full tunnel face examined during TBM cutterhead revision; bottom: cutterhead revision
Figure 3.1.8: Observation of the excavation process from the front of the TBM during excavation of the invert remaining from a rescue operation to release a TBM that was trapped for 5 months.

Figure 3.1.9: Operating the TBM during the performance of a start-up test.
### 3.1.2 Properties of TBMs Used for Data Collection

Data was collected at three different tunnel construction sites on 6 different TBMs, each site having two identical machines. The relevant TBM properties are listed in Table 3.1.1.

Table 3.1.1: Properties of TBMs used for data collection

<table>
<thead>
<tr>
<th>Manufacturer</th>
<th>Amsteg TBM</th>
<th>Lötschberg TBM</th>
<th>Bodio TBM</th>
</tr>
</thead>
<tbody>
<tr>
<td>Diameter</td>
<td>8.830 m</td>
<td>9.430 m</td>
<td>9.580 m</td>
</tr>
<tr>
<td>Cutter Diameter</td>
<td>17“ (432 mm)</td>
<td>17“ (432 mm)</td>
<td>17“ (432 mm)</td>
</tr>
<tr>
<td>Number of Cutters</td>
<td>60</td>
<td>58</td>
<td>62</td>
</tr>
<tr>
<td>Cutterhead Power</td>
<td>3.500 Kw</td>
<td>3.500 Kw</td>
<td>3.500 Kw</td>
</tr>
<tr>
<td>Cutterhead Rotation</td>
<td>Variable 0 – 6 RPM</td>
<td>Variable 0 – 6 RPM</td>
<td>Variable 0 – 6 RPM</td>
</tr>
</tbody>
</table>
3.2 TBM Performance Data

3.2.1 Introduction

In order to develop an understanding of the geological dependence of TBM excavation a machine performance analysis methodology was developed in which a variety of key TBM performance parameters obtained from the TBM data acquisition system (DAS) were selected for collection over long (300-500m) tunnel sections and correlated to the geology encountered. The parameters were selected based on their direct relation to the chipping process occurring at the TBM head and their ability, alone or in combination, to vary directly with the changing geology at the tunnel face. The parameters selected for collection were (see for example Figure 3.2.1):

1. penetration rate: depth of cutter penetration per rotation of cutterhead, mm/rev;
2. gross thrust: the force applied by the cutterhead, adjusted for friction loss on the cutterhead, in kN;
3. torque: the moment applied to the cutterhead to achieve rotation, in kNm;
4. rpm: the number of revolutions of the cutterhead per minute.

The corresponding geological information collected consisted of point load index strength testing of core samples at 1m intervals for a length of 400m, and a total of nine UCS tests on the same core. Other geological datasets were collected, as discussed in Chapter 4.

The data supplied by the DAS were collected at 10-second intervals. This data is often most useful when averaged over one stroke. All of the data have been averaged using specifically built macros to obtain single average values for each TBM stroke. This process is described in Appendix C.1.
Penetration rate describes the depth to which the cutters penetrate during each rotation of the cutterhead. This value is used, in conjunction with the rpm to determine the machine speed, in mm/min. The speed is not used in this research since it is not a basic unit and depends on both rpm and penetration rate, although the TBM DAS records speed by measuring the change in thrust piston extension over time, and rpm, to calculate the corresponding penetration rate. Although the penetration rate is not directly measured, it is a useful parameter as it relates directly to the chipping process.

Thrust describes the amount of force applied by the thrust pistons to the tunnel face. The DAS measures gross thrust, which includes losses due to friction, etc. The net thrust is calculated, as best as possible given the available data, by the procedure which is described in Appendix B.1. The net thrust is a better value for comparison between different TBMs than the gross thrust, although this can only be valid for machines with similar operation. The four machines used in this research were built by Herrenknecht AG with very similar designs (see Table 3.1.2) and can thus be compared confidently.
3.2.2.1 Start-up Test Penetration-Thrust Graphs

The start-up data, the first few data points taken during routine machine start-up or a denser suite of data obtained by deliberate and incremental machine acceleration, are capable of outlining the cutter-rock relationship in terms of rock mechanics by highlighting the grinding and chipping behaviour during excavation. This data can be used to interpret the geological conditions over a very short tunnel length (10s of cm). An intuitive method for displaying start-up test data is by graphing the penetration rate versus the net thrust.

A total of 16 start-up tests were performed on three different TBMs (the east and west TBMs used at Amsteg and the east TBM used in Bodio) in three different rock masses (Altkristallin, Southern Aar granite and Leventina gneiss). The start-up tests were undertaken by slowly and incrementally increasing the machine thrust from full stop to maximum thrust. By this methodology a denser dataset was collected and the shape of the start-up curve and the points of interest could be determined. The complete start-up test dataset is shown on a stoke basis plotted as penetration versus net (per cutter) thrust graphs in Figures 3.2.2-3.2.6. The Figures have been separated by TBM and the rock masses in which the tests were conducted, and will be interpreted in subsequent sections.

![Penetration-thrust graph with start-up data from Amsteg West machine in Altkristallin](image)

Figure 3.2.2: Penetration-thrust graph with start-up data from Amsteg West machine in Altkristallin (numbers indicate stroke number).
Figure 3.2.3: Penetration-thrust graph with start-up data from Amsteg East machine in Altkristallin.

Figure 3.2.4: Penetration-thrust graph with start-up data from Amsteg East machine in Southern Aar granite.
Figure 3.2.5: Penetration-thrust graph with start-up data from Amsteg East machine in Southern Aar granite.

Figure 3.2.6: Penetration-thrust graph with start-up data from Bodio East machine in Leventina gneiss.
3.2.3 Face Stress Condition

The face stress condition is the 3-dimensional stress state at the excavation surface of the tunnel face due to the combination of in-situ and induced stresses and was estimated using elastic 3-dimensional numerical models (see Figure 3.2.7). The input to the models was obtained by a combination of sparse in-situ stress measurements based on analysis of earthquake focal mechanisms: red is normal faulting, green is strike-slip faulting and blue is thrust faulting, as well as from borehole breakouts in black (Figure 3.2.7a) and the 3-dimensional topography and tectonic history of the area (3.2.7b – top) to develop a 3-dimensional stress model of the area (3.2.7b – centre). A slice through the model at the tunnel elevation reveals the estimated in-situ stress condition based on the topographic and tectonic model used (3.2.7b – bottom). The stress condition at the face was then estimated by creating a 3-dimensional tunnel oriented in the same direction as the tunnel of interest (3.2.7c).
Figure 3.2.7: Schematic representation of process by which face stress is estimated using information about topography and regional stress (from World Stress Map, 2005) (a), building a 3-D numerical stress model with the tunnel overlaid (b), leading to a 3-D elastic numerical stress model at the tunnel face (c)
This methodology was used to obtain an estimate of the stress condition encountered in the Gotthard tunnel excavated in the Altkristallin and Southern Aar granite rock. A topographic map was used to create a three-dimensional model of the topography in the area around the tunnel (Figure 3.2.8) in Examine 3-D (Rocscience Inc., 2005), from which the pre-tunnelling stresses were estimated (Figure 3.2.9). The pre-tunnelling stresses were used as input into a three-dimensional model of the tunnel in Examine 3-D (Rocscience Inc., 2005) to determine the tunnel-induced stress tensors 0.5km apart in the Altkristallin (Figure 3.2.10, left) and in the Southern Aar granite (3.2.10, right), and are summarised in Table 3.2.1.

Table 3.2.1: Summary of estimated in-situ stress conditions for two-cutter modelling

<table>
<thead>
<tr>
<th>Location</th>
<th>S1</th>
<th>S2</th>
<th>S3</th>
<th>Face Parallel</th>
<th>Face Perpendicular</th>
<th>Out-of-plane</th>
</tr>
</thead>
<tbody>
<tr>
<td>Current</td>
<td>42</td>
<td>36</td>
<td>28</td>
<td>38</td>
<td>33</td>
<td>35</td>
</tr>
<tr>
<td>Current-5</td>
<td>40</td>
<td>36</td>
<td>27</td>
<td>36</td>
<td>32</td>
<td>35</td>
</tr>
<tr>
<td>SAG</td>
<td>46-48</td>
<td>37</td>
<td>32</td>
<td>48</td>
<td>35</td>
<td>41</td>
</tr>
</tbody>
</table>
Figure 3.2.8: Examine 3-D model output of the topographic model showing the tunnel alignment and major principal stress at the tunnel elevation.

Figure 3.2.9: Pre-tunnel stress at the Gotthard tunnel depth showing locations at which tunnel-induced in-situ stresses are estimated in the Altkristallin and SAG (left: major principal stress; centre: intermediate principal stress; right: minor principal stress).
3.2.4 Rock Strength Data

Point load index testing (PLT) was performed on core samples collected parallel to the tunnel axis in the Southern Aar granite. The testing methodology and data processing are described in Appendix B.3. The core samples were tested in both the axial (tunnel face perpendicular) and diametral (tunnel face parallel) directions (Figure 3.2.11) and their values were averaged over 2m for easier comparison to the TBM data, which are also averaged over approximately 2m stroke lengths. The ratio of tunnel face parallel to perpendicular strength is the anisotropy index (AI) (Broch, 1983) and is plotted for the Southern Aar granite in Figure 3.2.12. The AI is 1 for isotropic rock, and the anisotropy is more pronounced the greater AI deviates from 1. The PLT data for the Southern Aar granite demonstrate that the geological anisotropy (schistosity) is also manifested as strength anisotropy (Figure 3.2.13) with foliation parallel values typically less than ½ to ¼ of the foliation perpendicular values.
Figure 3.2.11: Tunnel face perpendicular and parallel point load index strength data for Southern Aar granite.

Figure 3.2.12: Anisotropy Index data for Southern Aar granite.
Figure 3.2.13: Comparison of perpendicular and parallel point load index strength data for Southern Aar granite.

3.2.5 Tunnel Boundary and Face Instability

Overbreak at the tunnel boundary and tunnel face were recorded by the contractor at the Amsteg worksite. The tunnel wall overbreaks were recorded whenever they were observed, and are therefore a continuous record of overbreak depth and extent (Figure 3.2.14). Three additional points were added to the wall overbreak records corresponding to stress-related failure that occurred 20m behind the active tunnelling face. The tunnel face overbreaks were only recorded (Figure 3.2.15) twice per day, and therefore provide a relatively sparse dataset (Figure 3.2.16), but nevertheless provide a good indication of face condition. Photos of the tunnel face were collected (Figure 3.2.17) as permitted and correspond to the larger circles in Figure 3.2.16.
Figure 3.2.14: Maximum depth of tunnel wall overbreak based on wall records.

Figure 3.2.15: Sample depth of face overbreak records (courtesy AGN).
Figure 3.2.16: Maximum depth of face failure according to face stability records; larger circles correspond to photos of tunnel face in Figure 3.2.13.

Figure 3.2.17: Tunnel face photos; Tunnel metre locations on each photo. Scales variable, note 43cm cutters for scale. TBM head not in contact with rock, distance from face variable.
Figure 3.2.17 continued: Tunnel face photos; Tunnel metre locations on each photo. Scales variable, note 43cm cutters for scale. TBM head not in contact with rock, distance from face variable.
3.3 TBM Performance as an Indicator of Geological Conditions

3.3.1 Introduction

The TBM and geological data presented along a tunnel metre length are useful to identify changes in geology along the tunnel length, but in order to make interpretations about the impact of changing geological conditions on the TBM performance, the data must be plotted in combination with each other. The TBM performance analysis methodology introduced in Section 3.2 is based on the identification and collection of key parameters, as well as the combination of these parameters into meaningful graphs. The most important combinations are:

1. Penetration versus thrust graphs
2. Drillability index (DI), the ratio between penetration and thrust
3. Net advance rate (NAR), the ratio between active driving time and distance excavated
4. NAR versus DI graphs
5. Ratio between induced tunnel face stress and rock strength
6. NAR versus strength/stress ratio graphs

3.3.2 Penetration-Thrust Graphs and DI

Penetration-thrust graphs are employed to interpret the amount of force required to obtain penetration, and consist of plotting penetration rate (in mm/rev) versus total machine thrust (in kN). Two variants of this methodology are used: one based on average running data for an entire stroke (Figure 3.3.1), and the other on stroke start-up data (Figure 3.3.2).
Figure 3.3.1: Average stroke running data from Amsteg worksite in Southern Aar Granite showing locations of points and frequency distribution. The variation in points is related to variation in rock strength along the tunnel.

Figure 3.3.2: Schematic penetration versus thrust graph showing implied rock strengths based on TBM performance (points 1-6 described in Section 3.3.2.3).
3.3.2.1 Average Stroke Penetration-Thrust Graphs

The average stroke data, calculated as per Appendix 3.2, demonstrates the gross performance over each 2m tunnel length, and draws the relationship between the geological conditions and the TBM design limitations over large tunnel lengths. The penetration rate limit (arising from mucking limitations) and the gross thrust limit (arising from cutter limitations) both delineate the boundaries within which a TBM can safely be operated (dashed lines in Figure 3.3.2). Data points on the left of the graph are interpreted as representing favourable excavation conditions, with weaker rock corresponding to higher penetration rates and lower thrust requirements. Conversely, points on the right of the graph are interpreted as being unfavourable excavation conditions requiring higher thrust but with lower corresponding penetration.

3.3.2.2 Drillability Index

The location of either the average stroke running points or the start-up graphs for a particular rock type in penetration-thrust graphs can be used to delineate gross changes in geological conditions. A modification of the penetration and thrust data from Figure 3.2.1 is the ratio between penetration rate and thrust. This ratio is the slope of the line joining the origin and the plotted average stroke running point (4 in the graph in Figure 3.3.2). This ratio is termed the drillability index (DI) in this research, although it is also referred to as specific penetration (Rostami, Gertsch and Gertsch, 2002; Sapigni et al, 2002; Thuro and Plinninger, 2003), which is its inverse. The DI is a better indicator of the geological conditions than penetration rate only, since it is normalised by the applied thrust, on which it depends. Since the DI is a normalisation, a higher DI would suggest better excavation than a lower DI since a higher penetration rate is obtained with the same force (for example Figure 3.3.3). In this research, a higher DI value is interpreted as representing rock that is easier to excavate, either due to lower strength or preconditioning of the face through induced stresses. Conversely, a lower DI is interpreted as representing rock that is more difficult to excavate due to higher strength.
3.3.2.3 Start-up Test Penetration-Thrust Graphs

The start-up curves introduced in Section 3.2.2.1 are used as an indicator of the in-situ rock mass strength as it relates to TBM excavation. Curves on this graph display an inflection point, where the penetration rate increases more rapidly with increased application of thrust. With reference to the schematic in Figure 3.3.2, analysis of this graph consists of investigating geological and machine influences on: 1) initial penetration behaviour; 2) the location of the inflection point called critical penetration, which is hypothesised to represent the point at which grinding ceases and chipping commences; 3) the slope of the line past the inflection point; 4) machine/process related limitations to penetration and thrust; 5) hysteresis, which is the difference in behaviour between applying and relieving forces, in the start-up and slow-down portions of the graph; and 6) minimum thrust required to begin movement of the machine. The shape of the low-thrust portion of the graph (in Figure 3.3.2) in stable face conditions (1 and 6) may be dependent on machine behaviour or preconditioning of the tunnel face by stress. A probable cause of the inflection point (2) is a change in excavation process from grinding (creating fines) to chipping, and the location of this inflection point and the slope of the graph

Figure 3.3.3: Drillability index data for Southern Aar granite
beyond the inflection point (3) is related to mineralogical and fabric characteristics of the rock unit. Features 2 and 3 are considered to be critical indicators of cutter efficiency as controlled by hard rock properties. The material handling limit is controlled by conveyance, bucket design and revolution speed. The cutter thrust limit (4 in Figure 3.3.2) is controlled by cutter and TBM head design. The hysteresis (5 in Figure 3.3.2) between the start-up and slow-down portions of the curve is likely influenced by geological and friction-related mechanical factors.

The penetration-thrust graphs can only be used in the interpretation of the geological conditions if the assumption is made that the TBM is operated at either of the machine limits: the material handling limit or the cutter thrust limit. Otherwise, non-unique interpretations can be made of DI, which are not necessarily related to geology. In most cases the machine is operated near its limits to maximise production so this problem is considered rare (see Figure 3.2.1).

### 3.3.2.4 Quantification of Start-up Tests

A methodology developed by Bruland (1998) to investigate TBM performance, named penetration tests, greatly resembles the start-up tests. The Penetration tests are conducted by testing the TBM penetration at a series of thrust increments rather than a gradual increase in thrust, however, the output data are very similar. Bruland (1998) developed a methodology to quantify the results of the penetration test, which is applicable to the data from the start-up tests with minimal modification. The modified methodology is based on log (penetration) versus log (thrust) graphs through which linear fits are applied. The linear fit is associated to the following:

\[
\log_{10}(\text{penetration}) = A_R \log_{10}(\text{thrust}) + B_R
\]

where \( A_R \) and \( B_R \) are the constants arising from the linear regression. A unitless measure named the penetration coefficient, \( b \), was derived to be equal to the first regression constant, \( A_R \), by defining a basic penetration rate, \( i_o \), and relating them through the following (Bruland, 1998):

\[
i_o = \left( \frac{M_i}{M_1} \right)^b
\]

where \( M_i \) is the average thrust at each thrust increment and \( M_1 \) is the critical thrust required to achieve a penetration of 1mm/rev. Equation 3.3.2 is not directly applicable to the start-up tests because the basic penetration is the average penetration rate at each thrust interval. The derivation is, however, useful and results in this third equation:

\[
M_1 = 10^{\frac{A_R}{b}}
\]
The penetration coefficient and critical thrust could both be calculated using Equation 3.3.1 applied to the regression information from the start-up test data, as shown in Figures 3.3.4 to 3.3.8. The linear regressions are shown on each figure to demonstrate the source of the regression parameters. The quality of the regressions are acceptable with \( R^2 \) values between 0.63 and 0.97, with one stroke with an \( R^2 \) value of 0.31 due to the small number of datapoints.

\[
\begin{align*}
\text{Stroke 1729} & : y = 3.8803x - 6.6921, \quad R^2 = 0.6356 \\
\text{Stroke 1784} & : y = 2.5636x - 4.2569, \quad R^2 = 0.883 \\
\text{Stroke 1796} & : y = 2.6285x - 4.4082, \quad R^2 = 0.7707
\end{align*}
\]

Figure 3.3.4: Log Penetration - log thrust graph with start-up data from Amsteg East machine in Altkristallin
Figure 3.3.5 Log Penetration - log thrust graph with start-up data from Amsteg West machine in Altkristallin.

Figure 3.3.6: Log Penetration - log thrust graph with start-up data from Amsteg East machine in Southern Aar granite.
Figure 3.3.7: Log Penetration - log thrust graph with start-up data from Amsteg East machine in Southern Aar granite.

Figure 3.3.8: Log Penetration - log thrust graph with start-up data from Bodio West machine in Leventina gneiss.
In order to determine the appropriate critical thrust value, the inverse of the drillability index was plotted against penetration rate (Figures 3.3.9 to 3.3.11) and fit with power functions with $R^2$ values 0.92 to 0.99. The inflection point in this data demonstrates the penetration rate at which drillability increases dramatically. This point is interpreted as the point at which crushing gives way to chipping, similar to the inflection point in Figure 3.3.2. Graphing the data in this way provides a visual method by which the critical penetration rate is found, which for this dataset is taken as 1mm/rev, same as the value used by Bruland (1998). This value is used to calculate the critical thrust using Equation 3.3.3. The penetration coefficient and critical thrust are summarised in Table 3.3.1. The net cutter thrust was used in this analysis to allow the comparison of data from different TBMs.

![Graph](image)

Figure 3.3.9: Inverse drillability index versus penetration rate graph in Altkristallin showing power function regression lines
Figure 3.3.10: Inverse drillability index versus penetration rate graph in Southern Aar granite showing power function regression lines

Figure 3.3.11: Inverse drillability index versus penetration rate graph in Leventina gneiss showing power function regression lines
Table 3.3.1: Summary of TBM performance parameters obtained from start-up test analyses.

<table>
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<tr>
<th>Stroke</th>
<th>Penetration Coefficient $b=A_R$</th>
<th>$B_R$</th>
<th>Critical Net Thrust $M_1$ (kN)</th>
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<td>76</td>
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3.3.3 Impact of Face Instability on TBM Performance

3.3.3.1 Introduction to Face Instability

A complication arises in the analysis of TBM performance by penetration-thrust graphs in locations where unstable face conditions occur. These conditions lead to a different excavation process than a smooth stable face, but cannot be identified solely based on the penetration-thrust graphs. In unstable face conditions blocks are released from the face either due to pre-existing open joints or spalling in the face, rather than through the chipping process responsible for excavation of a stable face. In order to investigate the chipping process, locations in which unstable face conditions occurred during TBM excavation must be identified and analysed separately. In situations in which the face stress to strength ratio is low the machine must generate all of its own fractures at the face. There is a strength to stress ratio at which point the stresses are high enough or the rock strength is low enough to aid in fracture generation and the machine can run at its maximum speed. Beyond this point, with increasing face stress, the rock begins to fail considerably and blocks may be released. In order to spare the cutters and prevent an overload on the conveyor belt the TBM head rotation must be slowed and at times the machine must stop pushing to allow the removal of failed material from the face (Figure 3.3.12). This decreases the overall performance and results in longer stroke times, and indicators of TBM
Figure 3.3.12: Schematic of TBM progress through unstable face conditions cased by interaction between the induced tunnel face stress and fabric (top) or isotropic rock (bottom).

specifically related to face instability can be used to identify locations with unstable face conditions.

3.3.3.2 Net Advance Rate

A methodology for analysing the TBM performance data (penetration rate, thrust, torque and rpm) was developed to identify locations in which unstable face conditions likely occurred by developing a measure of TBM performance in zones with recorded face instability. In order to account for the loss of time arising from the need to clear the TBM head of rock blocks (not to be confused with normal utilisation: maintenance, support, cutter change, etc.) the TBM data were analysed for indicators of face instability. The main indicator of face failure is a dramatic increase in torque followed by the stoppage of thrust in combination with the continuation of TBM head rotation: Thrust = 0, RPM ≠ 0. The active driving time is, therefore, defined as the time during which the TBM head was turning, regardless of whether or not there was thrust (see Appendix 3.2 for full details). From this definition the net advance rate (NAR), which is the distance travelled by the TBM in one stroke divided by the amount of time the TBM was actively driving, was developed and calculated by macros for all TBM strokes (Figure 3.3.13).
NAR is best visualised in comparison with the geological condition at the tunnel face, since they are directly related, with NAR in the vertical axis and the ratio between the face stress condition and the strength of the rock on the horizontal axis. The NAR versus stress/strength ratio graph developed for the Southern Aar granite (Figure 3.3.14) is based on the face stress estimated as per Section 3.2.3 and the rock strength determined by point load index testing as per Section 3.2.4. A four-parameter log-normal function was fit through these data. The NAR is linearly spread while the stress/strength ratio, as a quotient can be displayed as a logarithmic value. Four parameters are necessary to introduce a y-intercept into the function since there are no NAR values at 0. The fits for face-perpendicular and parallel PLT strength are different in that the face-parallel data contains a nearly complete fit due to the high variability, including zero strength values in the face-parallel direction (parallel to foliation). The face-perpendicular data contains only the lower end of the curve since the strength in the perpendicular direction (perpendicular to foliation) was higher and never zero. The issue of foliation orientation with respect to the tunnel face and the cutter direction is apparent from this graph where depending on the orientation of foliation the excavation conditions could be in the lower, middle or upper sections of the graph, with different implications for excavation.

Figure 3.3.13: Net advance rate data for Southern Aar granite averaged over 2m intervals.
The schematic graph in Figure 3.3.15 shows that NAR is essentially a measure of the diminishing returns nature of changing stress/strength conditions at the tunnel face. A low stress/strength ratio arising from very high strength or low stress does not create a pre-conditioned tunnel face and leads to tough excavation and low NAR. A high stress/strength ratio arising from very high stress or low strength leads to face instability and low NAR. NAR is highest when the stress/strength ratio is optimised for pre-conditioning of the tunnel face that aids with chipping without causing spontaneous block formation.
3.3.3.3 Net Avance Rate and Drillability Index

The NAR value has non-unique interpretation, in that a low NAR value can be due to poor chipping in tough rock or face instability. In order to accurately identify face instability the NAR must be used in conjunction with other performance indicators. The data can be used in a NAR graph and compared to the rock strength/face stress ratio (as in Figure 3.3.14). In locations where these data are not available the DI value has been used in combination with the NAR value to differentiate between low NAR values arising from tough excavation conditions (correspondingly low DI) and arising from face instability (correspondingly high DI).

Figure 3.3.16 contains the NAR versus DI graph of data from the Leventina gneiss, Southern Aar granite and Central Aar granite in which the data were characterised according to recorded stable and unstable face conditions, as well as locations with very tough excavation conditions and poor chipping. NAR is theoretically limitless but in application a limit arises from the material handling limitations of the TBM. The strokes with stable face conditions have higher NAR than those with unstable face condition with a transitional NAR range between both face conditions.
conditions. The schematic in Figure 3.3.17 demonstrates the combinations of NAR and DI and the related interpretation of tunnel face conditions. In this figure the NAR is shown to have an upper limit and reductions due to poor chipping conditions or unstable face conditions. Points with stable face conditions are those with a linear NAR-DI relationship or high NAR, whereas those with unstable face conditions have low NAR values. Face instability can arise from different rock conditions such as spalling in tough to excavate (low DI) rock (i.e. Central Aar granite in Weh and Bertholet, 2005) and fabric induced fracture generation (i.e. Southern Aar granite).

Figure 3.3.16: NAR versus DI for Leventina gneiss, Southern Aar granite and Central Aar granite data.
3.3.4 Comparison of TBM Performance to Geological Characteristics and Tunnel Face Behaviour in the Southern Aar Granite

3.3.4.1 TBM Performance and Rock Strength

The point load test (PLT) data from Southern Aar Granite were compared to TBM performance represented by the net advance rate and the drillability index (Figure 3.3.18). Two sections are highlighted, each having different NAR, DI and PLT characteristics. The tunnel length indicated by the solid line is characterized by rock with a low point load strength index and well-defined foliation, corresponding to high NAR and high DI. The rock strength in this area is such that the TBM could chip efficiently through the rock, either due simply to low strength or favourable pre-conditioning of the face. It is likely that the stress induced in the tunnel face interacted with the well-defined foliation to induce small fractures in the rock, thereby aiding the chipping process and increasing performance.

The tunnel length indicated by the double dashed line is characterised by higher point load strength index, in more granitic rock where the foliation is not well defined and the mica content is lower, corresponding to low NAR and low DI. The rock strength in this area is such
Figure 3.3.18: Comparison between point load index strength and TBM net advance rate in Southern Aar granite; trendline for point load strength values is 3 point floating central average and for NAR is 15 floating central average; solid ellipse at left is low PLT and high NAR while double dashed ellipse at right is high PLT and low NAR
that efficient chipping was not occurring due to poor fracture generation and little or no pre-conditioning of the face by the induced stress.

In both of these locations the NAR and DI are directly related, suggesting that face instability did not adversely affect the TBM performance, and that chipping performance directly related to rock strength and pre-conditioning was responsible for TBM performance.

### 3.3.4.2 TBM Performance and Face Stability

Analyses of TBM performance and tunnel face behaviour in the Southern Aar Granite were also performed. Figure 3.3.19 presents a comparison of the depth of face failure and the net advance rate (NAR) as well as the drillability index (DI). The combination of these graphs highlights the interdependence of TBM performance in terms of advance rate and required thrust and allows the investigation of the effect of the condition of the tunnel face on TBM performance. Three areas are circled, corresponding to different combinations of NAR and DI, as well as different depth of face failure.

The double-dashed and solid lines in Figure 3.3.19 correspond to the lines in Figure 3.3.18, interpreted as having high and low face-parallel point load strength indeces, respectively. As no point load testing was done prior to tunnel metre 116425, an analogous relationship between NAR and point load index strength is not available for the TBM data prior to this point. Examination of the drill core in these locations, however, shows that a high occurrence of disking corresponds to locations circled with a dashed line in Figure 3.3.19, where the depth of face failure is elevated. Disking in the core can occur due to a reduction in rock strength or a change in stress conditions. A comparison of estimated stress condition at the two extremities of the core suggests that it does not change significantly along the length of the core boring. The disking observed in the core in this area would, therefore, likely be in part due to decreased rock strength, which would have been manifested as low face-parallel point load index strength.

The highlighted zones in Figure 3.3.19 demonstrate that NAR can be affected by poor chipping performance, as shown by the double dashed zone, as well as high depth of face failure, as shown by the single dashed zone. At depth of failure above 2m the NAR is reduced by face instability by the same magnitude as the zone with very poor chipping performance. Face depth of failure dominates NAR above approximately 0.75m, while chipping performance dominates NAR at DI below approximately 0.001, for the machine configuration under study.
Figure 3.3.19: Comparison between TBM net advance rate and drillability index (above) and depth of face failure (below) in Southern Aar granite. High depth of failure occurs at low NAR and high DI locations (dashed ellipse), moderate depth of failure occurs at moderate NAR and high DI, and no face failure occurs at low NAR and low DI.
3.3.4.3 TBM Performance as an Indicator of Geological and Face Condition

Figure 3.3.20 presents a comparison of NAR and the depth of face failure. The large points correlate with the zones highlighted in Figure 3.3.19 and the depth of face failure highlighted in Figure 3.3.21. The triangles, at low depth of face failure, have variable NAR resulting from changes in condition unrelated to depth of face failure, but rather related to chipping performance. The diamonds, at moderate depth of face failure, have high NAR values with variability around an average NAR of approximately 28±3 mm/min, implying that under these conditions the TBM performance is assisted by the conditions in the face that lead to moderate failure. The squares, at high depth of face failure, have NAR points that vary around an average of approximately 22±4 mm/min, indicating that the conditions at the face adversely affect TBM performance.

Figure 3.3.20: Graph showing effect of face failure on net advance rate in Southern Aar granite; fit is a third-order polynomial. Point shapes correspond to zones in Figure 3.3.19 (squares = dashed, diamonds = solid and triangles = double dashed).
Figure 3.3.21: Depth of face failure in Southern Aar granite with points categorised according to face stability.

Figure 3.3.22 presents a comparison of the NAR and the drillability index (DI), both calculated as average values of all stroke data. Some points also correlate with the areas circled in Figure 3.3.19 and the depth of face failure highlighted in Figure 3.3.21. The low DI portion (triangles) on this graph corresponds to stable tunnel face conditions and can be interpreted as being tough to excavate (as discussed earlier; see Figure 3.3.2) with a high point load index strength (Figure 3.3.23), likely because the rock was strong enough to withstand the induced face stress. The photo at TM 116619 in Figure 3.3.24 shows a stable face with clearly visible kerfs, which indicate that the cutters were being used to generate chips and/or fines, and the penetration rate was limited by the amount of thrust that can be applied to the cutters. A linear to slightly non-linear relationship exists in this area between DI and NAR, in which lower strength rock results in easier excavation up to a maximum, dictated by the material handling capacity of the TBM, as discussed (Figure 3.3.17).
Figure 3.3.22: Comparison graph between NAR and DI for Southern Aar granite. Point shapes correspond to circles in Figure 3.3.19 (squares = dashed, diamonds = solid and triangles = double dashed).

Figure 3.3.23: Point load index test results for Southern Aar granite with points categorised according to face stability.
The higher DI portions of the graph in Figure 3.3.22 can be broken into two groups corresponding to positive and negative impact on NAR of increasing depth of face failure. The areas with diamonds that positively affect NAR correspond to rocks with lower depth of face failure in Figure 3.3.21 and low to medium point load index strength in Figure 3.3.23. The photos from TM 116475, 116574 and 116679 in Figure 3.3.24 show examples of the face condition with failure around one metre deep. In these conditions the face is sufficiently stressed relative to rock strength such that thrust requirements to excavate the rock in the damaged portion of the face are reduced but the negative effects associated with the need to clear broken material from the face have less impact on the net advance rate compared to the positive effects of a preconditioned face.

The areas in Figure 3.3.22 with squares where NAR is negatively affected correspond to the areas having higher depth of face failure highlighted in Figure 3.3.21. The face photos in Figure 3.3.24 corresponding to tunnel metres 116281-116321 show considerable face spalling; in the case of tunnel metre 116281 to the depth of 2.5m. In this area the face stress caused deep failure such that the TBM was only required to cut the gauge area, resulting in lower thrust requirements due to the smaller disk contact area and increased drillability. For the machine being studied face stability can be considered as depth of face failure less than 0.75 m and instability can be considered as depth of face failure greater than 2 m, with a transition in terms of impact on NAR from increasing face instability and depth of failure. In more general terms, the thresholds for face stability and instability can be stated as approximately 10% and 20% of the tunnel diameter, respectively, with a transition in between.

Figure 3.3.24: Tunnel face photos; Tunnel metre locations on each photo. Scales variable, note 43cm cutters for scale. TBM head not in contact with rock, distance from face variable.
Figure 3.3.24 continued: Tunnel face photos; Tunnel metre locations on each photo. Scales variable, note 43cm cutters for scale. TBM head not in contact with rock, distance from face variable.
3.3.5 Chipping Performance Designation

3.3.5.1 TBM Performance as an Indication of Chipping Performance

The chipping performance of the rock for which F-Factors were collected was determined by examining the penetration versus thrust graphs and designating rocks that are sensitive to spalling, in terms of chip generation and at the tunnel face, during TBM excavation. Only rocks with face-perpendicular PLT values above 2.5 MPa were used, since weak rocks insensitive to spalling could plot at the same location on the penetration versus thrust graph as stronger rocks that are sensitive to spalling. Some samples tested had face-perpendicular PLT values below 2.5 MPa, but were obtained from disked core. Core disking arises from the development of tensile stress when the principal and intermediate stresses are perpendicular to the axis of the core (Kaga, Matsuki and Sakaguchi, 2003) and is, therefore, a spalling-type phenomenon. No direct relationship has been found, however between disking in the core and face instability, suggesting that disking alone is not a suitable indicator of face instability. Figure 3.3.25 is similar to Figure 3.3.2, but modified to demonstrate the chipping performance designation with respect to TBM performance, as well as implied rock strength from TBM performance.

Figure 3.3.25: Schematic penetration versus thrust graph showing equivalent rock strengths and regions of chipping performance.
The zones in Figure 3.3.26 are also used to classify rocks as non-chipping (stroke data falling in the lower linear portion), and chipping (strokes with higher DI), with the distinction for strokes in which unstable face conditions occurred. This is complementary to Figure 3.3.25, and plotting strokes in this manner provides the ability to distinguish strokes in which the penetration was high due to good chipping conditions or due to face pre-conditioning by moderate instability. Both graphs are necessary to confirm the interpretation of chipping performance for individual strokes, in particular in transition areas both in terms of strength and face stability.

3.3.5.2 Chipping versus Non-Chipping Classification

The methodology to classify strokes as chipping or non-chipping is based on the classification of the start-up tests performed in the Altkristallin and the Southern Aar granite, and the examination of the other corresponding indicators (NAR, DI) from each stroke for which start-up tests are available.

Figures 3.3.27 and 3.3.28 show all start-up test data from the tests performed with the TBMs used in Amsteg (Altkristallin and Southern Aar granite) and Bodio (Leventina gneiss), respectively, as in Figure 3.3.2. Stroke 4420 is of interest because its start-up test data falls in the middle of the dataset, but its point load strength (see Table 3.3.2) is much lower due to severe discing of the core (Figure 3.3.29). The sample from stroke 4420 was taken from the Southern Aar granite but its location on the graph in Figure 3.3.27 is within the Altkristallin range.
Figure 3.3.27: Start-up test data from all tests conducted on TBMs used at Amsteg. Strokes 1477 to 1796 in Altkristallin and strokes 4387-4493 in Southern Aar granite.

Figure 3.3.28: Start-up test data from all tests conducted on TBM used at Bodio in Leventina Gneiss.
Table 3.3.2: Performance and strength summary for start-up test strokes in Altkristallin and Southern Aar granite shown in Figure 3.3.13. Values in italics have been calculated using relationship: UCS=24·PLT. Measured UCS taken from nearby tests conducted by contractor, Courtesy AGN.

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<th>Stroke</th>
<th>Tunnel Metre</th>
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<th>NAR (mm/min)</th>
<th>PLT face-perp. (MPa)</th>
<th>UCS calculated (MPa)</th>
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<td>9.04E-04</td>
<td>28.8</td>
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<td>126.5</td>
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<td>5.63E-04</td>
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<td>8.0</td>
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<td>5.0</td>
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</table>

Figure 3.3.29: Photograph of core sample corresponding to stroke 4420 in Southern Aar granite showing discing.

Figure 3.3.30 is a NAR versus net DI graph (DI normalized to number of cutters to allow comparison of multiple TBMs) of the start-up test dataset using the same scale as in Figure 3.3.16. A comparison of the two graphs, and the schematic in Figure 3.3.26 shows that the majority of the start-up tests were conducted in zones of stable face conditions, however samples taken from strokes 1796, 1784 and 1729 show progressively more signs of the negative impact on
performance from face instability. Samples 4387-4397 and 4477-4493 were taken from the Southern Aar granite, although none of these showed signs of diskin, but rather showed signs of increasing difficulty in excavation in Figures 3.3.27 and 3.3.30 due to poor chipping. Table 3.3.3 shows the chipping performance assigned to each stroke in the start-up test dataset.

Figure 3.3.30: NAR versus DI graph for start-up test dataset, symbols correspond to symbols in Figures 3.3.13 and 3.3.14.

Table 3.3.3: Chipping performance and face stability condition for strokes from the start-up dataset.

<table>
<thead>
<tr>
<th>Stroke</th>
<th>Chipping Performance</th>
<th>Face Stability</th>
</tr>
</thead>
<tbody>
<tr>
<td>1477</td>
<td>Chipping</td>
<td>Stable</td>
</tr>
<tr>
<td>1503</td>
<td>Chipping</td>
<td>Stable</td>
</tr>
<tr>
<td>1516</td>
<td>Chipping</td>
<td>Slightly unstable</td>
</tr>
<tr>
<td>1729</td>
<td>Chipping</td>
<td>Moderately unstable</td>
</tr>
<tr>
<td>1784</td>
<td>Chipping</td>
<td>Slightly unstable</td>
</tr>
<tr>
<td>1796</td>
<td>Chipping</td>
<td>Slightly unstable</td>
</tr>
<tr>
<td>4387</td>
<td>Non-Chipping</td>
<td>Stable</td>
</tr>
<tr>
<td>4397</td>
<td>Non-Chipping</td>
<td>Stable</td>
</tr>
<tr>
<td>4420</td>
<td>Chipping</td>
<td>Stable</td>
</tr>
<tr>
<td>4477</td>
<td>Non-Chipping</td>
<td>Stable</td>
</tr>
<tr>
<td>4478</td>
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<td>Stable</td>
</tr>
<tr>
<td>4489</td>
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<tr>
<td>4493</td>
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<td>Stable</td>
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<tr>
<td>869</td>
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<td>Stable</td>
</tr>
<tr>
<td>876</td>
<td>Non-Chipping</td>
<td>Stable</td>
</tr>
<tr>
<td>877</td>
<td>Non-Chipping</td>
<td>Stable</td>
</tr>
</tbody>
</table>
The penetration coefficient and critical net thrust for a penetration rate (introduced in Section 3.3.2.4) of 1mm/rev determined by analysis of the start-up tests in Section 3.3.2.4 were used to verify the chipping performance and tunnel face stability designations in Table 3.3.3. The penetration coefficient and critical net thrust show trends with chipping performance and tunnel face instability when categorised as such and plotted versus DI and NAR, although the regressions on all data points and data points separated according to chipping performance do not provide additional support for the chipping performance categorisation. For penetration coefficient this trend is a positive relationship with DI and NAR (Figures 3.3.31 and 3.3.32) for chipping rocks, as was established in Figure 3.3.25, while for non-chipping rocks it is a negative relationship. For critical net thrust it is a negative relationship with DI (Figure 3.3.33) for all TBM excavation conditions and a positive relationship with NAR (Figure 3.3.34) for chipping rocks and a negative relationship for non-chipping rocks. Neither the penetration coefficient nor the critical net thrust further confirm the use of DI or NAR for chipping performance or tunnel face stability due to conflicting relationships and non-unique values.

Figure 3.3.31: Penetration coefficient versus the net drillability index categorised according to chipping performance and tunnel face stability
Figure 3.3.32: Penetration coefficient versus NAR categorised according to chipping performance and tunnel face stability.

Figure 3.3.33: Critical net thrust versus the net drillability index categorised according to chipping performance and tunnel face stability.
The penetration coefficient and critical net thrust were plotted versus each other to determine their applicability to chipping performance and tunnel face stability verification (Figure 3.3.35). With this approach it was possible to identify penetration coefficient versus critical net thrust zones in which the chipping performance and tunnel face stability designations are clearly defined. The lines defining the zones represent the ratio between critical net thrust and penetration coefficient, with slopes of approximately 1/36 and 1/18, which was plotted against NAR and DI (Figure 3.3.36). Again zones were defined for chipping performance and tunnel face stability, since the ratio having negative linear relationships with both NAR and DI have $R^2$ values of 0.48 and 0.68, respectively, which are too low to confidently plot them in this fashion. Both figures, by using zones, further verify the categorisation of the start-up test data according to chipping performance and tunnel face stability.
Figure 3.3.35: Penetration coefficient versus critical net thrust graph categorised according to chipping performance and tunnel face stability.

Figure 3.3.36: Critical net thrust / penetration coefficient ratio versus NAR and DI, categorised according to chipping performance and tunnel face stability.
3.3.5.3 Receiver/Response Operating Characteristic (ROC) Curves

The Receiver/Response Operating Characteristic (ROC) curve process was developed for signal analysis in aviation in the 1940s but has since been very successfully applied to medical science as a way to analyse the effectiveness of test diagnoses for health conditions (Sackett et al, 1991). It is a method by which indicators for a condition can be analysed to determine whether or not they are good indicators for the condition and at which threshold a positive diagnosis is most likely. This can be applied to any situation in which a “diagnosis” is made based on results from a “test”. In this research, the condition being diagnosed is chipping performance (chipping versus non-chipping rocks) and the tests being analysed are indicators of chipping performance.

Manually creating a ROC curve for an indicator requires the following general steps (Sackett et al, 1991):
1. Select several test thresholds for the indicator
2. Calculate the sensitivity and specificity at each threshold
3. Plot the sensitivity versus 1 minus specificity (1-specificity) (Figure 3.3.37)
4. Determine the best threshold value by finding the threshold which results in the greatest number of true positives and lowest number of false negatives
5. Determine the area under the ROC curve where 1 is a perfect relationship (Figure 3.3.37 A) and 0.5 denotes no relationship (Figure 3.3.37 D)

The sensitivity is calculated as:
\[ sensitivity = \frac{a}{a + c} \]  \hspace{1cm} 3.3.4

where a is the number of positive diagnoses below or at a certain threshold and c is the number of positive diagnoses above a certain threshold.

The specificity is calculated as:
\[ specificity = \frac{d}{b + d} \]  \hspace{1cm} 3.3.5

where d is the number of negative diagnoses above a certain threshold and b is the number of negative diagnoses below or at a certain threshold.

The number of true positives and false negatives are represented by the likelihood ratio:
\[ LR^+ = \frac{sensitivity}{1-specificity} \]  \hspace{1cm} 3.3.6
\[ LR^- = \frac{1 - sensitivity}{specificity} \]  \hspace{1cm} 3.3.7
The software Sigmaplot (Systat, 2006) has a built-in function to calculate ROC curves and output the likelihood ratios and histograms of the data. ROC curves were created for each of the F-Factors collected by thin section analysis, and classified as chipping or non-chipping based on TBM performance data. For this research the following are used to evaluate the goodness of prediction based on the ROC curve area:

- .90-1 = excellent
- .80-.90 = good
- .70-.80 = fair
- .60-.70 = poor
- .50-.60 = fail

### 3.3.5.4 ROC Curve Analysis of Start-up Test Verification of Chipping Performance and Tunnel Face Stability Designation

To further verify the categorisation of the start-up tests according to chipping performance and tunnel face stability, ROC curves were generated with the data from Table 3.3.1 and the ratio of critical net thrust and penetration coefficient, as shown in Figures 3.3.38 and 3.3.39. The area under both curves are 1, showing that the chipping performance and tunnel face stability designations are perfectly verified with start-up test data, further supporting the methodology for defining these categories using TBM performance data as described in Section 3.3.5.2. The ratio of critical net thrust to penetration coefficient thresholds for the stress and geological conditions under which the Southern Aar granite (SAG) was excavated determined by this analysis are as follows:

- Good chipping with face instability: below 18kN
- Good chipping with stable face: between 18kN and 36kN
- Poor chipping with stable face: above 36kN
Figure 3.3.38: ROC curve for verification of chipping performance using the ratio of critical net thrust to penetration coefficient.

Figure 3.3.39: ROC curve for verification of tunnel face stability using the ratio of critical net thrust to penetration coefficient.
3.3.5.5 Rule Set for Chipping Performance Classification

The selection of whether or not a sample was considered chipping or non-chipping using TBM performance data was based on the following rules:

- Non-chipping is based on a combination of the location of the sample in the graph in Figure 3.3.16 below 22 mm/min on the NAR axis and below 0.00075 mm/min-kN on the DI axis, as well as its location in the lower right end of the average running penetration versus thrust zone in Figure 3.3.1 or 3.3.2.

- Chipping is based on a combination of the location of the sample on the graph in Figure 3.3.16 above 28 mm/min on the NAR axis and above 0.00075 mm/min-kN on the DI axis, as well as its location in the upper portion of the average running penetration versus thrust zone in Figure 3.3.1 or 3.3.2.

- Chipping – with face instability is based on a combination of the location of the sample on the graph in Figure 3.3.16 below 28 mm/min on the NAR axis and above 0.00075 mm/min-kN on the DI axis, as well as its location in the upper left end of the average running penetration versus thrust zone in Figure 3.3.1 or 3.3.2.

- The penetration coefficient and critical net thrust cannot be used for strokes without start-up test data due to the lack of sufficient data at low penetration rate – low thrust that is used to define the linear regression necessary to determine penetration coefficient and critical net thrust. This is demonstrated in Figure 3.3.40 in which start-up data and all stroke data for stroke 1477 were fit with linear regressions. The fit to all the stroke data does not coincide with the start-up test data, demonstrating that stroke data is not applicable to this methodology, which can only be used to verify the start-up test data at this point, but should give thresholds for start-up data.
3.3.6 Point Load Index Strength Analysis

The point load index strength data described in Section 3.2.4 were analysed to determine the relationship between rock strength and geological characteristics. The values were separated into samples with and without fabric. The samples without fabric were compared to mineralogy and grain size, whereas the samples with fabric were compared to fabric type and intensity.

3.3.6.1 Sample Without Fabric

Increased mica content leads to lower point load index strength, and the trend is most pronounced for samples without fabric (Figure 3.3.41). Increased quartz to feldspar ratio leads to higher point load index strength, with a more pronounced trend for samples without fabric (Figure 3.3.42). The grain size has a parabolic relationship with point load index strength, again with a more pronounced relationship for samples without fabric (Figure 3.3.43). These data show that more complex factors impact point load index strength when samples have fabric, but relationships are clearly defined for samples without fabric, in particular the relationships with mica content and grain size.
Figure 3.3.41: Comparison of mica content and point load test index strength for all samples and samples without fabric highlighted.

Figure 3.3.42: Comparison of quartz to feldspar ratio and point load test index strength for all samples and samples without fabric highlighted.
3.3.6.2 Samples With Fabric

Fabric type and intensity are not easily quantified, and they are compared to point load index strength according to fabric type and intensity categories. The fabric types are in increasing order of intensity, as are the subcategories related to decreasing microlithon spacing (Figure 3.3.44). For all fabric types the trends are more defined for face-parallel PLT index strength than for face-perpendicular, likely because the fabric is parallel to the face parallel direction allowing for more influence from the fabric. For rocks with mineral preferred orientation (MPO), increasing microlithon spacing leads to decreasing point load index strength. The opposite is true for rocks with cleavage, while rocks with schistosity do not provide a clear relationship. This is likely due to the differences in the nature of the types of schistosity, which are not only dependent on microlithon spacing, but also on type of microlithon. The increasing strength with decreased microlithon spacing for rock with mineral preferred orientation likely arises from the slightly smaller grain size and smaller microlithon size for the rocks with narrowly spaced minerals with preferred orientation. The decrease in point load index strength for rocks with cleavage with decreasing microlithon spacing is likely to do with the increasing mica content associated with
higher fabric intensity (Figure 3.3.45), in addition to the ease of fracture initiation along weaker cleavage planes, which would have a greater impact at smaller spacing. The decreased microlithon spacing is also manifested as smaller grain size, the effect of which is counteracted by the higher density of weaker cleavage planes.

Figure 3.3.44: Comparison of fabric type and intensity with point load test index strength for all samples with fabric (MPO is mineral preferred orientation).
3.3.7 Laboratory Strength and Domain Classification to Predict Chipping Performance and Face Stability

3.3.7.1 Chipping Performance and Laboratory Strength

The point load index strength data were processed using ROC curves to investigate the relationship between point load index strength and chipping performance. The face-perpendicular strength (Figure 3.3.46) and face-parallel strength (Figure 3.3.47) both show good relationships to chipping performance. The threshold face-perpendicular PLT index strength at which good chipping is reduced to poor chipping is approximately 6MPa while the threshold for face-parallel PLT index strength is 2MPa. The face-parallel PLT index strength is a better predictor of chipping performance than face-perpendicular PLT index strength because the face-parallel direction tests the ability of rock to fracture in a direction that is parallel to the tunnel face, which is the direction through which face-parallel fractures propagate to generate chips. These results show that PLT index strength tests are a good predictor of chipping performance under the conditions encountered during excavation of the SAG, and that the test samples should be oriented to coincide with the tunnel face such that sample failure is induced in the same direction as fractures leading to chipping at the tunnel face. The PLT was not a good predictor
for face stability, however, since the interactions between strength, induced stress and fabric are perhaps too complex for this type of analysis.

Figure 3.3.46: ROC curve for investigation of relationship between face-perpendicular PLT and chipping performance.

Figure 3.3.47: ROC curve for investigation of relationship between face-parallel PLT and chipping performance.
3.3.7.2 Chipping Performance and Geological Domain Classification

A ROC analysis was undertaken to determine the relationship between geological domain classification described in Section 2.3.2.3 and chipping performance (Figure 3.3.48) and face stability (Figure 3.3.49). The geological domains are excellent predictors of chipping performance and good predictors of tunnel face stability. The rock curves were used to assign numbers to each domain (A-J), in the order that provided the best ROC curve area. The domain values were ordered in increasing order with decreasing chipping performance, where values 8 and lower (domains B, C, D, E, F, H, I and J) have good chipping and greater than 8 (domains A and G) have poor chipping. A second set of domain values were ordered with increasing face instability, where domains with values 5 and 6 (domains A and D) have mild face instability, domains with values 7 and 8 (domains G and H) have moderate face instability, and domains with values 9 and 10 (domains B and C) have considerable face instability, while all domains below 4 (domains E, F, I and J) are stable at the face. As such, the domains are not assigned the same values when they are used to predict chipping performance as when they are used to predict tunnel face stability. These domains can be used to identify similar characteristics, and use these to extrapolate which characteristics lead to improved chipping and face stability. The domains themselves, and their associated values, however, should not be used in and of themselves for extrapolation to other projects. This analysis shows that even simple rock type classification into simple domains based on meso-geological characteristics such as mineralogy, grain size, fabric and metre-scale variability in terms of geological characteristics and discontinuities can be very helpful to identify domains in which chipping and/or face stability issues could arise.
Figure 3.3.48: ROC curve for investigation of relationship between geological domains and chipping performance.

Figure 3.3.49: ROC curve for investigation of relationship between geological domains and tunnel face stability.
3.3.7.3 Chipping Performance and TBM Domain Classification

The TBM data were classified into domains comprising tunnel lengths with similar TBM behaviour. Figure 3.3.50 shows typical relationships between drillability index (DI) and the net advance rate (NAR). With respect to Figure 3.3.50: (1) represents low penetrability resulting in low net advance rate arising from tough excavation in a solid face; (2) represents improved penetrability and net advance rate arising from easy penetration in a solid face; (3) represents low penetrability but resulting high net advance rate; (4) represents variable, opposing moderate penetrability and net advance rate trends; (5) represents high penetrability but low resulting net advance rate, due to face instability.

A ROC analysis was undertaken to determine the ability of TBM domain classification to relate to chipping performance (Figure 3.3.51) and tunnel face stability (Figure 3.3.52). The TBM domain classification is an excellent predictor of chipping performance and a fair predictor of tunnel face stability. The domain values were used as such since they were originally designated with chipping performance in mind with increasing chipping performance correlated with increasing domain value. This type of classification cannot be used prior to excavation, but demonstrates a second methodology by which TBM data can be classified according to chipping performance using NAR and DI if start-up tests are unavailable.

Figure 3.3.50: TBM performance relationships between the drillability index (DI) and the net advance rate (NAR), TM is tunnel metre.
Figure 3.3.51: ROC curve for investigation of relationship between TBM domains and chipping performance.

Figure 3.3.52: ROC curve for investigation of relationship between TBM domains and tunnel face stability.
3.3.7.4 Domain Classification and Laboratory Strength

The domain values described in Section 3.3.7.2 were compared to point load index strength values. The geological domain values ordered with decreasing chipping performance show a clear positive relationship with PLT index strength (Figure 3.3.53). The geological domain values ordered with increasing face instability show a poor relationship with PLT index strength (Figure 3.3.54). What is interesting is that the PLT index strength decreases with increasing face instability within each of the categories (mild instability, mixed stability and high instability). The TBM domains show a negative relationship with increasing domain value and PLT index strength (Figure 3.3.55).

This analysis shows that geological characteristics can be classified into meaningful domains that correlate well with laboratory strength, although the relationship between the domain values ordered according to face instability is more complex, likely due to the complex process involved in inducing face instability that cannot be solely characterized using laboratory strength testing. This further supports the use of PLT index strength to characterize samples for excavation design, especially when used in conjunction with geological characterisation.
Figure 3.3.53: Geological domains ordered according to chipping performance compared to point load index strength.
Figure 3.3.54: Geological domains ordered according to tunnel face stability compared to point load index strength.

Figure 3.3.55: TBM domains compared to point load index strength.
3.3.7.5 Domain Classification and Geological Characteristics

The geological domains were compared to geological characteristics collected using thin section analysis. The geological domains ordered according to chipping performance, as in Figure 3.3.52 have decreasing mica content and increasing quartz to feldspar ratio with increasing domain value (Figure 3.3.56), and decreasing grain size with decreasing domain value (Figure 3.3.57). This again shows that the domains can be used to meaningfully classify the rocks.

Figure 3.3.56: Geological domains compared to mica content and quartz to feldspar ratio.
Figure 3.3.57: Geological domains compared to grain size.
3.4 Summary of TBM Excavation of the Swiss Alpine Base Tunnels

3.4.1 TBM Data Collection in the Alpine Base Tunnels

TBM performance data were collected using a data acquisition system built into the Herrenknecht TBM control system. The key parameters that were used in analysis were the penetration rate, gross thrust, torque and RPM. In order to obtain a detailed dataset of parameters over the full range of penetration and thrust, start-up tests were developed in which the TBM operator slowly increased the TBM thrust after a full stop. In addition to TBM performance data, geological data were collected in the form of rock samples, tunnel wall and face maps, photos and detailed descriptions, rock mass characterisation and rock core point load strength testing.

3.4.2 TBM Performance Data

The TBM performance data were processed to obtain average values assigned to each TBM stroke. The TBM performance data were also processed such that they could be geographically related to the geological and strength testing data according to the location along the tunnel length at which the data were collected. This allowed the data to be analysed in terms of the TBM performance, and compared to the geological conditions under which they were collected.

3.4.3 TBM Performance as an Indicator of Geological Conditions

The TBM performance data were analysed to determine the impact of the intact rock conditions at the face as well as stress induced failure. A process at the tunnel face was described in which the interaction of the induced stress condition, the intact rock strength, spalling sensitivity and fabric type and orientation led to induced face instability through originally massive rock. The net advance rate was defined to identify the strokes at which face instability occurred. The net advance rate is determined by dividing the stroke length by the active driving time, which is the total time during which the TBM head was actively turning. This ratio takes
into account the negative impact on advance rate due to the need to remove fallen unstable tunnel face material during tunnelling by TBM cutterhead rotation without applied thrust.

The analysis and quantification of start-up test data, strength testing data and average stroke running data were used to develop a methodology by which chipping performance and induced face instability could be categorised. This methodology was used to categorize the location at which each rock sample used for geological analysis was collected into chipping versus non-chipping and stable versus unstable tunnel face.

The PLT index strength test data, geological domains and TBM performance domains were used to investigate their ability to differentiate chipping performance and tunnel face stability categories. It was found that PLT index strength test data were an excellent predictor of chipping performance in which rocks with axial strength below 6MPa and face-parallel strength below 2MPa exhibited good chipping performance. The face-parallel strength was a better predictor due to the orientation of the induced fracture parallel to the fracture orientation during chipping. These thresholds hold true for the in-situ stress conditions encountered during excavation of the Southern Aar granite, and it is not possible to say if these thresholds can be extrapolated to other conditions. Point load testing should be further investigated for its ability to predict chipping performance under other excavation conditions.

The geological domains were shown to provide meaningful characterisation of the geological characteristics. They were also shown to differentiate between chipping performance categories and tunnel face stability. Domains A and G (see Section 2.3.2.3 and Appendix B.4) were found to exhibit poor chipping. Their common characteristics are low variability >10m scale, low density of shear zones and fractures, low intensity fabric and no fabric, low mica content and high quartz to feldspar ratio. The common characteristics of the samples with good chipping are increased fabric intensity, higher shear zone density (3-5m spacing), and higher mica content. All other characteristics vary between domains.

The geological domains most associated with face instability were domains B and C, although domains A, D, G and H also had occurrences of face instability (see Section 2.3.2.3 and Appendix B.4). Domains B and C typically have high mica content, and the majority of the samples have cleavage fabric, which is composed of oriented weakness planes, even though the rock is massive, and tends to break more easily along these planes of weakness. In the SAG the fabric tends to be oriented between 0 and 21° to the tunnel face, which likely facilitates face semi-parallel failure leading to face instability.

The domain classifications were found to relate well to point load index strength data, demonstrating that the domains are meaningful with respect to rock strength. They were also
found to relate to geological characteristics, showing that they can capture changes in geology. These analyses show that a simple classification of rock domains at the 100’s of metre scale using field mapping, core logging, or information from previous tunnelling can provide a method by which domains at high risk for poor chipping and/or face instability can be identified and the extent of these domains can be delineated to determine if their impact on chipping performance and face stability will be critical to the project. This should be undertaken early in the excavation design stage so that high risk for poor chipping or face instability domains are identified and more detailed investigation can be undertaken to determine the nature of the impact the geology will have on chipping performance and face stability.
Chapter 4: Micromechanics and Rock Behaviour

4.1 Fundamentals of Rock Fracture Mechanics

4.1.1 Introduction

Three fundamental modes of fracture propagation have been documented as (Ceriolo and Di Tommaso, 1998): opening Mode I, sliding Mode II and tearing Mode III (Figure 4.1.1). Modes I and II are of particular relevance to rock mechanics and can be simplified to extensile (Mode I) and shear (Mode II) failure. It is also possible for mixed I and II mode failure to occur, which is a combination of tensile and shear failure (Chang, Lee and Jeon, 2002; Jian-An and Sijing, 1985; Schultz, 2000). Of particular interest to this research is the creation of fractures under compressive stress, especially within the low confinement realm that is generally present at rock excavation boundaries. Both extensile and shear failure have been shown to occur under compressive stress (Diederichs, 2003) and the relationship between the occurrence and sensitivities of each mode and fracture behaviour during rock yielding is one of the main goals of this research. In particular, identifying the rock characteristics that promote extensile failure under compressive stress and the eventual failure of the rock are of interest, with the particular application to chipping and tunnel face stability during TBM excavation is important. Within this framework, the fundamentals of rock fracture mechanics will be summarized, with a particular focus on extensile fracture propagation and blunting under compressive stress.

Figure 4.1.1: Schematic showing the three modes of failure (from Diederichs, 1999).
4.1.1.1 Extensile Fracture under Compressive Stress

Griffith (1921; 1924) first proposed the tensile extension of pre-existing fractures under a compressive stress field. His work in glass rods laid the groundwork for understanding what has since been loosely and generally termed brittle behaviour. A semantic and theoretical discussion of the meaning of brittleness is contained in Section 4.1.2. The principle of extensile fracture propagation within a compressive stress field has become the foundation of rock mechanics, in particular for crystalline rock. The following is a short summary of the primary equations for extensile crack propagation under a compressive stress field.

4.1.1.1.1 Griffith’s Failure Criterion

Minimum tensile stress, $\sigma_t$, for crack extension in plane strain applied normal to the fracture axis (Griffith, 1924) can be defined as:

$$\sigma_t = \sqrt{\frac{2ET}{\pi c}}$$

where $E$ is the Young’s Modulus, $T$ is the surface tension and $c$ is the fracture half-length. This demonstrates that the tensile strength will depend on the stiffness of the material and the inverse square root of the flaw size.

This was extended to different triaxial stress fields in which sliding of the fracture can be predicted according to (Griffith, 1924):

If $3\sigma_3 + \sigma_1 < 0$, then failure will occur if $\sigma_3 = K = \text{tensile strength}$.

If $3\sigma_3 + \sigma_1 > 0$, then failure will occur if $\left(\sigma_1 - \sigma_3\right)^2 - 8K(\sigma_1 + \sigma_3) = 0$, where $K = \text{tensile strength}$. For uniaxial compression conditions: $\sigma_3 = 0$ and $\sigma_1 = \sigma_c = -8\sigma_t$, becomes the logical conclusions of these formulas and conditions (Brace, 1960).

The key is that based on the Young’s Modulus and the surface tension of a crack of a certain length, the tensile stress necessary to continue to pull that crack apart can be calculated. In addition, the maximum tensile stress present at the crack tip can also be calculated. From this, the theoretical compressive strength can be found based on the tensile strength if an average or minimum crack dimension or flaw size is assumed for the inherent internal defects within a real material. Rocks were found to have a range of tensile to compressive strength from 1:7 to 1:11 (Griffith, 1924), although the relationship between the uniaxial compressive strength and tensile strength is more complicated and depends on rock failure behaviour arising from differences in characteristics.
4.1.1.2 Application of Griffith’s Failure Criterion to Rocks

Re-examination of Griffith’s theory and attempts to apply it to the failure of rocks has led to modifications specifically for rock fracture (Brace, 1960). The major change arises from the assumption that pre-existing fractures will initially close under compression, the surfaces of which can then slide against each other during deformation. This necessitates the introduction of a friction coefficient to account for the force required to slide two frictional surfaces past each other. This allowed the reconciliation of Griffith’s theory of fracture extensions with the Mohr – Coulomb theory of rock strength envelopes and resulted in the definition of a modified Griffith theory (Brace, 1960):

\[ \tau = 2K + \mu \sigma \]  

where \( \tau \) is the shear stress component, \( K \) is the tensile strength (which is assumed to be \( 1/8 \sigma_c \)), \( 2K \) is termed \( \tau_0 \) or cohesion (Brace, 1960), \( \mu \) is the friction coefficient (\( \mu = \tan \phi \)), and \( \sigma \) is the normal component of stress.

Brace (1960) also introduces the idea that friction and cohesion do not likely exist together, where cohesion has dropped to zero after sliding begins, at which point friction is activated.

Detailed investigations of the stress versus strain behaviour of rocks under compressive loading identified the processes occurring during rock failure. Brace et al. (1966) identified four regions in stress-strain graphs during which different processes were occurring:

I: Elastic behaviour.
II: Elastic behaviour in which grains deform and shift slightly, and pre-existing microfractures close, but damage is recoverable. Rock volume decreases.
III: Frictional sliding and deformation of grains. Volumetric strain reversal and increase in rock volume arising from opening of axial cracks near closed pre-existing crack ends and formation of new open axial fractures.
IV: Sample rupture.

It was found (Brace et al., 1966) that the load at which the sample ruptured was many times higher than the load at which new fractures initiated. It was discussed that in rocks, as in other brittle material, the longest fractures grow first, and smaller cracks do not begin to grow until the longest cracks have become stable and additional stress is applied. Brace et al. (1966) conclude that Griffith’s theory cannot apply to rock failure under compression since the rupture stress is much higher than the stress at which new fractures are initiated. They postulate that sample rupture under compression occurs by faulting, not by growth of a critically oriented pre-existing
crack, as proposed by Griffith, and suggest that the critically oriented pre-existing crack likely becomes stable at stresses near \( c' \), the stress at which new cracking begins.

4.1.1.3 Concluding Remarks

Examination of the interaction of pre-existing fractures and the applied stresses during sample failure have led to conclusions regarding the role of microfractures. Pre-existing microfractures, as well as cleavage and grain boundaries play a large role in new fracture initiation and propagation (Kranz, 1983; Nasseri, Mohanty and Prasad, 2002; Nasseri, Mohanty and Robin, 2005; Tapponier and Brace, 1976), as was predicted and discussed by Griffith (1924). The majority of new microcracks in samples loaded under compression were found to be extensional (Kranz, 1983; Moore and Lockner, 1995; Tapponier and Brace, 1976), suggesting that extensile fracture initiation and propagation should be focussed on in the examination of rock yielding in sample or at excavation boundaries. Rock failure depends on the localisation of the induced micro-fractures into faults, or macro-fractures, through the destruction of intact rock bridges (Kuksenko et al, 1996; Lockner and Madden, 1991; Schultz, 2000; Wong, 1982). Initiation, propagation as well as interaction are, therefore, critical to the behaviour of brittle samples under compression and should form an integral part of the description of brittle failure.

4.1.2 Britteness

4.1.2.1 Introduction

When crystalline rocks are described as being brittle many definitions could be used, including those based on compressive versus tensile strength (Andreev, 1995), those based on the stress strain curves (Bishop, 1967; Kahraman, 2002) and those based on the relationship between lab strength and in-situ strength (Diederichs et al., 2004). Previous work by Diederichs et al. (2004) suggests that different combinations of rock characteristics are responsible for strength magnitude, and the ratio between laboratory sample strength and in-situ excavation boundary strength.

Several definitions for brittleness exist, for example (Andreev, 1995):

- “The brittle material/rock lacks the property of ductility” (Hetenyi, 1966; Morley, 1944)
- “Brittle material/rock usually terminate by fracture at or only slightly beyond the yield stress” (Obert and Duvall, 1967).
• “The material/rock is said to be brittle if the internal cohesion of rock materials, which are deforming in their elastic range is broken” (Ramsay, 1967).

• “Brittleness is defined as the property of material/rock that fractures with little or no plastic deformation” (geologists’ definition, anonymous, 1960).

Starting as far back as 1966 researchers have endeavoured to define brittleness using indices, based on laboratory testing or empirical methods. Some indices characterise brittleness in terms of the stress-strain curve while others define it in terms of strength envelopes. This has resulted in the creation of numerous indices, some of which are described below. In the following discussion the various brittleness indices are defined as B with a subscript. The subscripts are numbered sequentially based on when they are introduced.

### 4.1.2.2 Indices Based on Stress or Strain

Bishop (1967) created a brittleness index, $B_1$, based on the post-peak behaviour of a rock, as seen in a stress-strain curve.

$$B_1 = \frac{\tau_f - \tau_r}{\tau_f}$$

where $\tau_f = $ peak shear strength, and $\tau_r = $ residual shear strength. His formulation gives an indication of the brittle versus plastic behaviour that can be explained by the rapid loss of strength of a sample (mostly due to cohesion loss) compared to a sample where the post-peak strength is the same as the peak strength due to the mostly frictional behaviour of the substance, respectively. This behaviour (Figure 4.1.2) is commonly used to describe brittle rocks, but is dependent on sample size and loading rate, and, therefore, could not be used as a universal index for brittleness.
The two following indices are based on the mechanical behaviour of rock, as seen in a stress-strain curve (Andreev, 1995).

\[ B_2 = \varepsilon_{ii} \times 100\% \]  

where \( \varepsilon_{ii} \) = irrecoverable longitudinal strain (see Figure 4.1.3).

\[ B_3 = \frac{\varepsilon_{1e}}{\varepsilon_{lt}} \]  

where \( \varepsilon_{1e} \) = elastic longitudinal strain, and \( \varepsilon_{lt} \) = total longitudinal strain. These two indices serve as indicators of brittleness versus plasticity based on the fact that plastic rocks will have a greater amount of plastic strain before reaching their peak strength than will more brittle ones. These three indices may be useful as indicators but they say nothing about the underlying processes leading to the behaviour of the rocks in question.

### 4.1.2.3 Indices Based on Strength Envelopes

All of the following indices are based on the observed trend that rocks exhibiting more brittle behaviour have higher \( \sigma_c/\sigma_t \) ratios. The term brittleness here refers to rocks that have been observed to fail in a brittle fashion, which is to quickly lose strength after failure has occurred. The indices are formulated as follows (Andreev, 1995):
\[ B_4 = \frac{\sigma_c}{\sigma_t} \]  \hspace{1cm} (4.1.6)

where \( \sigma_c \) = uniaxial compressive strength and \( \sigma_t \) = tensile strength

\[ B_5 = \frac{\sigma_c \times \sigma_t}{2} \]  \hspace{1cm} (4.1.7)

\[ B_6 = \frac{\sigma_c - \sigma_t}{\sigma_c + \sigma_t} \]  \hspace{1cm} (4.1.8)

\[ B_7 = q \sigma_c \]  \hspace{1cm} (4.1.9)

where \( q \) = % fines from the Protodyakonov impact test. Kahraman (2002) found that the brittleness of rocks as predicted by equations \( B_4 \) and \( B_6 \) correlated exceptionally well with each other. This would also be expected from \( B_5 \), \( B_4 \) and \( B_6 \) since all three are based on the same two factors: the rock’s uniaxial compressive strength and its tensile strength.

The last formulation, \( B_7 \) was not found to correlate (Kahraman, 2002). \( B_7 \) includes the \( q \) parameter based on observations during the Protodyakonov impact test that rocks exhibiting brittle behaviour tend to create more fines than those exhibiting plastic behaviour. The more brittle rock will incur microscopic fractures, upon impact, creating fines, while the more plastic rock will simply indent upon impact.

These brittleness indices are a result of the greater likelihood of a rock with lower tensile strength to fail in a brittle way, i.e. by tensile fracture, than a rock with a higher tensile strength. That is, the rock with the lower tensile strength will be more likely to initiate extensile fractures that can propagate through the rock than the other, thus exhibiting brittle behaviour, such as spalling.

These indices, like the three previous indices, are only indicators of behaviour and again are not descriptors of the behaviour of the rock, although it must be noted that Kahraman (2002) found that a correlation exists between brittleness as described by \( B_4 \) and \( B_6 \) and the penetration rate of a TBM. The validity of these findings is not clear, however, as the rocks tested were sedimentary, not crystalline, in which case the reader may question whether the rocks tested can really be defined as brittle rocks. Certainly in consideration of the crystalline rocks that are of interest to this research, these sedimentary rocks do not offer a suitable comparison.
4.1.2.4 Index Based on Fracture Initiation

This index differs from the other indices in that it does incorporate some form of description of the behaviour of the rock. It is formulated as follows (Andreev, 1995):

$$B_s = \frac{\sigma_{il}}{R_m}$$  \hspace{1cm} (4.1.10)

where $\sigma_{il}$ = damage initiation stress, and $R_m$ = ultimate peak strength. This index describes the difference in stress magnitude between the initiation of micro fractures and the eventual failure of the rock. The damage initiation stress may be found using acoustic emissions during uniaxial compression strength testing.

Figure 4.1.4 shows the location of the damage initiation line in a conceptual model in $\sigma_1$-$\sigma_3$ space. As can be seen, the point at which fracture initiation begins is lower than the actual failure point, for a constant confining stress. This index is not a direct representation of brittleness, however. It is, in fact, an indicator of the lower bound for strength (spalling). Above the damage initiation stress (labelled in Figure 4.1.4), fractures begin to form. This is the onset of either crushing mechanisms or spalling mechanisms. Under the appropriate conditions of low $\sigma_3/\sigma_1$ ratios, spalling will dominate and brittle failure will ensue. Current research (and indeed

![Diagram](image)

Figure 4.1.4: Conceptual model of various rock behaviour envelopes in principal stress space (after Diederichs, 2003).
this project) involves the repeatable assessment of the confining stress ratio within which the
strength drops to this lower bound (critical confinement limit in Figure 4.1.4). This confinement
limit, hereafter referred to as the Spalling Limit, is discussed later in this document.

4.1.2.5 Index Based on Separation of Cohesion and Friction

4.1.2.5.1 Separation of Cohesion and Friction

Before the final brittleness index is presented, the rock mechanics principles on which
this index is based must be described. The final index is based on the separation of cohesion and
friction when describing a rock’s failure and post-peak behaviour.

As described in Section 4.1.2.2, brittleness is commonly described as the rapid loss of
strength once failure has occurred. This is demonstrated in Figures 4.1.5 and 4.1.6. The first
figure shows the typical strength envelopes of a brittle rock, while the second figure shows a
stress-strain curve corresponding to the rock represented in the previous figure. Following the
loading paths, at 1 the rock is undergoing some loading force, which increases until it reaches the
peak strength envelope, at 2. At this point, the rock strength rapidly drops to the residual
strength, shown by number 3.

Figure 4.1.5: Strength envelopes for typical brittle rock behaviour characterisation.
In this typical case there is a drop in cohesion from peak to residual and there may or may not be a change in friction angle from peak to residual (in this example the friction angle is assumed to remain the same).

Recent work by Martin (1997) has shown that during the failure of rock the mobilisation of friction is not concurrent with the loss of the frictional component of strength. In simple terms this can be described in a few steps:

1. The rock is initially held together by cohesion, and friction has no bearing on strength because there are no surfaces on which friction can act, or no confinement to provide the frictional strength component.
2. Once some micro fracturing begins to occur, some of the cohesion within the rock is lost.
3. As more fracturing occurs and the rock begins to deform, the frictional strength component begins to be mobilised on those fracture surfaces that have been pressed together by confinement and are sliding past each other as deformation is occurring.
4. Once the sample has failed (i.e. post-peak) most of the cohesion may be lost and the strength of the rock is largely due to the mobilised frictional strength component.

This behaviour can be represented on a schematic of cohesion and friction mobilisation versus strain (Figure 4.1.7). This figure shows that with increasing strain on a sample (i.e. deformation of the sample during failure) the effect of cohesion (confinement independent strength) is lost while the effect of friction (confinement dependent strength) becomes dominant.
This behaviour can be demonstrated in terms of Mohr-Coulomb strength envelopes as shown in Figure 4.1.8, where the peak strength envelope is specified with peak cohesion and a near 0 degree friction angle, while the residual strength envelope is specified with reduced cohesion and the peak friction angle. This characterisation of the behaviour of rock has been useful for input parameter selection for numerical modelling.

It must be noted, however, that this model is a phenomenological model. That is, it produces the correct bilinear failure envelope expected in spalling conditions but does not represent true rock mechanics (Kaiser, 2007). As a parameter for the description of rock behaviour the apparent cohesion strength is really a function of the tensile strength of the rock, while the apparent friction strength is due to the elastic generation of extension strain and tensile crack accumulation (Diederichs, 2003).

![Figure 4.1.7: Schematic showing cohesion loss and friction mobilisation with increased strain on a failing rock sample (Martin, 1997).](image)

Figure 4.1.7: Schematic showing cohesion loss and friction mobilisation with increased strain on a failing rock sample (Martin, 1997).
Figure 4.1.8 shows that the behaviour of a rock with these strength envelopes is highly
dependent on confinement. At low confinement the rock will behave in a brittle fashion,
displaying strain softening post-peak, while at high confinement the rock will behave in a more
plastic behaviour, displaying strain hardening behaviour. Clearly, in tunnel excavation we are
only concerned with the first scenario, while the second scenario would be more applicable to
tectonic processes.

4.1.2.5.2 Index Description

The information presented in the previous section forms the basis for the final brittleness
index (Hajidabdolmahid and Kaiser, 2002). This index aims to quantify brittleness in terms of
the rate of cohesion loss versus friction mobilisation while a sample is failing:

\[ B_q = I_{B_e} = \frac{\varepsilon^p_i - \varepsilon^p_c}{\varepsilon^p_c} \]  \hspace{1cm} 4.1.11

where \( \varepsilon^p_c \) = strain at peak cohesion mobilisation, and \( \varepsilon^p_i \) = strain at peak frictional strength
component mobilisation. Figure 4.1.9 shows the relationship between the two strain values used
in \( B_q \) on a stress-strain curve. This index is representative of the lag strain between the two peak
strength values, where the greater the lag, the more brittle the rock since in brittle rocks failure of
the rock occurs before the frictional strength component can be mobilised. One might ask,
however, if this is truly a rock property or simply a function of confinement since the frictional
strength component is dependent on confinement.
4.1.3 Spalling limit

Before any discussion of spalling can proceed, the environment in which this type of failure occurs must be understood. The boundaries of an excavation, including the face of a tunnel, are environments of low confining stress. This is important because rock behaviour, in particular brittle rock behaviour, is highly dependent on confinement. It is also necessary to demonstrate the difference in stress conditions between an excavation boundary and a UCS test (Figure 4.1.10). The shapes of cylinders used in testing result in the creation of hoop stresses along the boundaries, increasing confinement, and therefore increasing the peak strength of the sample (Diederichs et al., 2004). At the boundary of an excavation, however, such conditions do not exist, meaning that any fracture that can propagate will not be resisted by confinement and may lead to spalling failure at the boundary at lower excavation boundary stress than would be predicted by the UCS value of the rock.
Figure 4.1.10: Left: unrestricted crack propagation at excavation boundary; right: crack suppression due to induced hoop stresses in cylindrical samples (modified from Diederichs, 2003).

The geomechanical theory behind the definition of the spalling limit is based on the idea of brittle (spalling) behaviour being a result of heterogeneity at low confinement (Diederichs, 2003). The principle of crack initiation was discussed briefly in Sections 4.1.1.3.2 and 4.1.2.4. Figure 4.1.4 shows conceptually where the fracture initiation curve occurs in a principal stress space. This figure also shows where the fracture interaction curve occurs, and is important because it is at this point that rock failure actually occurs. Another important curve is the critical confinement curve, which is related to spalling behaviour in Diederichs et al. (2004).

The behaviour of hard rocks at low confinement is conceptualised in Figure 4.1.11. This figure shows the composite strength curve, composed of segments from the lower bound strength (crack initiation) at low confinements, the upper bound strength (fracture interaction) at high confinements, connected by a transition related to the spalling limit curve. It is important to note that excavation boundaries are located in the area of the graph denoted by spalling failure, axial splitting and tensile failure due to low confinement.
The lower bound and upper bound curves are dependent on a variety of factors and will not be described here. Of importance to this discussion are the factors on which the spalling limit is dependent, since a rock subjected to stresses above the initiation threshold, and to the left of the spalling limit has the potential for failure below the fracture interaction threshold due to unstable crack propagation (Diederichs et al., 2004).

The spalling limit, represented as a ratio of $\sigma_3/\sigma_1$, is dependent on the degree of heterogeneity and external factors, such as stress rotation and rock damage. In particular an increase in heterogeneity, high damage levels and unfavourable stress conditions will result in a shallower spalling limit curve, thus allowing for greater potential for spalling failure (Diederichs, 2003). Defining this limit in a testable configuration is of great interest to this research.

The rock characteristics and in-situ stress condition will together result in crack promotion or blunting, thus affecting the relationship between excavation boundary strength and laboratory UCS, which is reduced due to brittle behaviour, as opposed to comparability of UCS to excavation boundary strength for rocks that exhibit non-brittle behaviour (Diederichs et al., 2004).

While all of the relationships between factors leading to brittle behaviour under this model have not been fully quantified, nor have they been fully related to mechanical excavation,
it is believed that a discussion related to this topic will prove useful for the advancement of the research and the identification of courses for investigation.

4.1.4 Summary of Rock Fracture Mechanics as Applied to Excavation

- As stated in Kahraman (2002), there is no uniformity between brittleness indices and each should be used separately, as required by the situation.
- For the purpose of mechanical excavations of tunnels, where behaviour at the excavation boundary is of utmost importance, current brittleness index formulations are not capable of adequately quantifying the brittleness of the rock or predicting its behaviour.
- The concept of cohesion and frictional strength separation is a useful one for simulating the lowering of the rock strength at low confinement, but is not a true representation of the behaviour of the rock.
- At low confinement, determining the location of onset of failure is critical because this may represent the failure threshold of the rock.
- Britteness is dependent on factors such as heterogeneity, rock damage and stress conditions.
- These factors form the basis of the investigations into the quantification of brittle behaviour for engineering works near excavation boundaries. In particular, the behaviour of individual crystals (in terms of crushing versus splitting, for example) and their relationships with neighbouring crystals within a heterogeneous matrix is the key to characterisation of brittle behaviour of rock.
- The goal is to relate the mechanical behaviour of brittle rock in the low confinement range to the mineralogy and fabric itself.
4.2 Geological Characterisation for Spall Sensitivity

4.2.1 Introduction to Geomechanical Characterisation

A methodology for presenting geological data in a way in which it can be classified into a series of modification factors for lab yield strength, herein referred to as F Factors, was developed. Based on an extensive literature review of fundamental rock mechanics principles and fracture theory, as well as laboratory and field investigation findings by other authors (in particular, Diederichs et al., 2004; Riedmüller and Schubert, 2001), three factors were identified for their importance to rock yield behaviour at excavation boundaries.

The F Factors, outlined in Table 4.2.1, were developed as part of the overall methodology to relate geological characteristics to rock mechanics demonstrated in Figure 4.2.1. This flow chart illustrates the geological characteristic associated with each F Factor and how these factors are interpreted to provide geomechanical information for determination of spall sensitivity and fracture potential. The term spall sensitivity, $F_{SS}$, is used to describe the impact that mineralogy, grain size and foliation have on rock yield behaviour that could lead to sudden spalling at the excavation boundary, as described in Section 3.1. The spall sensitivity factor, combined with standard lab strength predictions, describes differences in rock yield behaviour under induced stress conditions during TBM excavation. For this purpose, $F_{TF}$ is used to describe the potential for fracture at the cutter scale and the entire tunnel face scale.

This methodology can be used to better anticipate rock yield behaviour at excavation boundaries to make better predictions for TBM performance. This system was created for crystalline rocks, not sedimentary rocks, due to the difference in characteristics of grain boundaries found in sedimentary rocks. Ultramafic, volcanic and highly altered ore rocks are also not considered due to limited data in these rock types. The following sections describe the development of the classification scheme from Figure 4.2.1 as a conceptual model. Section 4.3 will address the quantification and calibration of the classification scheme.
Table 4.2.1. Description of geomechanical characterisation factors

<table>
<thead>
<tr>
<th>Factor</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>( F_M )</td>
<td>Mineralogy. Total and relative percentage of major minerals, ( F_{MM} ), such as quartz, olivine, feldspar, calcite, amphibole, and pyroxene, and total and relative percentage of accessory minerals, ( F_{MA} ), such as biotite, muscovite, garnet, pyrite and magnetite, are weighted for their low, medium or high impact on fracturing and spalling behaviour. The combination of the two results in a low, medium or high designation for the mineralogy factor.</td>
</tr>
<tr>
<td>( F_G )</td>
<td>Grain size and grain size distribution. Median grain size, ( F_{GP} ), grain size reduction due to tectonic processes, such as subgrain formation and grain boundary migration, ( F_{GT} ), and grain size distribution, primary or secondary resulting from tectonic deformation, ( F_{GD} ), are designated low, medium or high, and are combined to result in a low, medium or high designation for grain size and grain size distribution impact on fracturing and spalling behaviour.</td>
</tr>
<tr>
<td>( F_A )</td>
<td>Anisotropy. Foliation type, ( F_{AF} ), and foliation dimension, ( F_{AD} ), in combination as ( F_A ), are assigned a low, medium or high designation for impact on fracturing and spalling behaviour.</td>
</tr>
<tr>
<td>( F_{SS} )</td>
<td>Spall Sensitivity. ( F_M ), ( F_G ) and ( F_A ) are combined to determine the low, medium or high sensitivity to isotropic spalling.</td>
</tr>
<tr>
<td>( F_{FI} )</td>
<td>Fracture Potential. Standard lab strength values and ( F_{SS} ), for isotropic rocks are combined to determine the fracture potential of the rock, and normally manifests itself as a reduction of the lab strength value, representing excavation boundary strength: ( F_{FI} = F_{SS} \times \text{lab strength} ).</td>
</tr>
</tbody>
</table>

Data Collection

<table>
<thead>
<tr>
<th>Sample Data</th>
<th>( F ) Factors</th>
<th>Spalling Potential</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mineralogy</td>
<td>( F_{MM} \rightarrow F_M )</td>
<td>( F_{SS} \rightarrow F_{FI} )</td>
</tr>
<tr>
<td>Grain Size and Grain Size Distribution</td>
<td>( F_{GP} \rightarrow F_G )</td>
<td></td>
</tr>
<tr>
<td>Fabric - Anisotropy</td>
<td>( F_{AF} \rightarrow F_A )</td>
<td></td>
</tr>
</tbody>
</table>

![F factors schematic showing data collection, classification and combination to obtain fracture potential. Legend: \( F_{MM} \) – mineralogy major; \( F_{MA} \) – mineralogy accessory; \( F_M \) – mineralogy; \( F_{GP} \) – grain size petrological; \( F_{GT} \) – grain size tectonic; \( F_{GD} \) – grain size distribution; \( F_G \) – grain size and grain size distribution; \( F_{AF} \) – fabric type; \( F_{AD} \) – fabric scale; \( F_A \) – anisotropy; \( F_{SS} \) – spalling sensitivity; \( F_{FI} \) – fracture potential.](image-url)
4.2.2 Mineralogy Classification – $F_M$

4.2.2.1 Geomechanical Characterisation of Mineralogy

Determination of mineralogy is a key procedure when characterising rocks, be it for geological or engineering purposes. For the most part, mineralogical composition forms the basis for geological classification. The content of certain minerals has been used to predict the behaviour of different rock types for engineering applications such as drilling rate prediction (Thuro, 1997, for example), tunnel and mine design (Diederichs et al., 2004; Riedmüller and Schubert, 2001, for example), and TBM cutterhead design (for example (Sapigni et al, 2002)). Work presented in other studies strongly suggests that mineralogy plays a crucial role in rock yield strength and yield behaviour. A need, therefore, exists for a method to translate mineral content of a rock into useable parameters for input into the geomechanical classification scheme outlined in Figure 4.2.1.

The proposed method for disseminating mineralogical and rock type data for engineering purposes is based on an evaluation of what role the most common crystalline rock forming minerals play in rock behaviour. The minerals are grouped according to their impact on fracture initiation and propagation, and therefore on brittle yield. Two main groups of minerals were identified and are referred to in this research as major minerals, $F_{MM}$, and accessory minerals, $F_{MA}$. The major mineral category includes minerals with moderately high stiffness and strength, and which can be thought of as fracture receptors. The accessory mineral category includes minerals on both ends of the stiffness and strength spectrums and can be considered as fracture initiators; they are also typically, but not necessarily, less abundant than the major minerals. The results of the categorisation of the rock based on its major and accessory mineral abundances are used to determine the mineralogy factor, $F_M$ (Figure 4.2.1), categorised according to its impact on yield behaviour: low, medium or high. The following literature review discusses the findings of investigations into mineral content versus rock yield strength and yield behaviour.

4.2.2.1.1 Importance of Mineralogy to Rock Yield Strength and Yield Behaviour

Geologists have extensively studied the role that mineralogy plays in a rock’s rheology, in addition to textures and internal structures (Davis and Reynolds, 1996). Of particular interest are the differences in strength between quartz and feldspar at different pressure and temperature conditions. Feldspar has been shown to be stronger than quartz under low temperature conditions.
(Passchier and Trouw, 1996). Early work by Wawersik and Brace (1971) showed that there was value in differentiating the behaviour of different minerals and their interactions during the rock yielding process. They found that in Westerly granite 35% of new fractures were generated along grain boundaries and 65% through quartz and feldspar, demonstrating the importance of mineral properties for rock yield strength.

Several studies have investigated the impact of mica content in particular on the yield strength and yield behaviour of rock. Schist and gneiss samples loaded perpendicular to the fabric have been shown to exhibit decreasing yield strength with increasing mica content and undergo brittle yield behaviour up to a mica content of about 30% and a transition to steady state shear at a mica content of about 50% (Shea and Kronenberg, 1993). For samples loaded at 45 degrees to the fabric increasing mica content had a less distinct impact on strength, where peak strength was reached at a mica content of about 20%, and the transition to stable ductile shear occurred at a mica content of about 40% (Shea and Kronenberg, 1993). Although mica content itself was found to have no direct impact on fragmentation resistance during laboratory crushing for aggregate production, it was found to be an important parameter when it forms foliation planes that define weakness planes and thus decrease fragmentation resistance (Åkesson et al, 2003). Relationships between fabric parameters and rock characteristics (i.e. lab yield strength characteristics such as compressive and tensile strength, physical characteristics such as heat capacity, etc.) were found to be slightly different when mica content was taken into account (Büchi, 1988).

Other studies have examined how the different common rock forming minerals affect the yield strength and yield behaviour of rock (i.e. Howarth and Rowlands, 1987; Tuğrul and Zarif, 1999; Singh, Singh and Singh, 2001). The lab strength of the granites and gneisses is directly related to mineral grain size, especially when compared to the grain size of quartz and mica, the strongest and weakest phases, respectively (Prikryl, 2001). In Westerly granite minor mineral constituents may play a role that is as important as, or more important than, the major mineral constituents because of their differences in elastic, plastic or rheological properties (Tapponier and Brace, 1976).
4.2.2.2 Mineral Impact on Fracture

4.2.2.2.1 Investigations of Mineralogical Controls on Fracture Characteristics

Most investigations into the impact of individual mineral types on fracture behaviour were conducted in granite. Observations of crack progression during compression of Westerly granite were reported by Wong (1982) and Tapponnier and Brace (1976) and are summarised as follows:

1. At low stress, grain boundary healed cracks open first (Tapponnier and Brace, 1976)
2. High angle transgranular cracks occur in feldspars and biotite, corresponding to cleavage, but are absent in quartz (Wong, 1982)
3. Magnetite is frequently cracked throughout the entire stress history (Tapponnier and Brace, 1976)
4. At or near peak stress micas show kinking and contribute to large conspicuous cracks in the specimen (Tapponnier and Brace, 1976)
5. Quartz grains appear shattered, feldspar grains contain many fractures and biotites are kinked in the post peak region (Wong, 1982)
6. Biotite can act as a crack arrestor when cleavage is oriented at a large angle to the propagating fracture (Tapponnier and Brace, 1976)
7. Local tensile stress induced at the interface of two grains with different elastic or plastic properties could be a main cause of induced fracturing (Tapponnier and Brace, 1976)

A similar investigation in Hong Kong granites was presented in Li (2001) and Li et al. (2003), and the following conclusions were made:

1. Grain boundary cracks are initiated first, especially quartz-quartz grain boundaries oriented parallel to loading.
2. Micas often show signs of yielding and may play a major role in crack initiation in nearby quartz grains.
3. New cracks form in quartz at ~0.5σc and propagate parallel to the direction of loading.
4. New cracks form in plagioclase at ~0.75σc parallel to the direction of loading but with zigzag patterns aligned with the cleavage and/or connecting pores, suggesting that cleavage and pores play a major role in crack development and arresting in plagioclase.
5. Cracks will propagate through quartz boundaries, they may or may not propagate through feldspar boundaries, but their growth rate will be reduced, and cracks are usually halted at biotite boundaries.

6. In the post-peak region, feldspars show signs of sudden yielding.

The dissemination of the role of quartz and feldspars in the strength of rock based on fracture behaviour is a complex process; nevertheless, a few authors have concluded that feldspar tends to contribute to lowering the yield strength of granitic rock. Quartz in Lac du Bonnet Granite is stronger than feldspar and encircles feldspar and biotite, likely giving granite its strength (Lajtai, 1998). Feldspar plays an important role in strength reduction, perhaps because of its cleavage and microfissures (Tuğrul and Zarif, 1999). Lab yield strength (UCS, tensile, point load index) versus mineral content shows a negative relationship between feldspar content and strength, and a positive relationship between both quartz content (Figure 4.2.2a) and quartz to feldspar ratio versus strength (Figure 4.2.3a). The strength ratios show an increase in PLT/Tensile and Tensile/UCS but no change in PLT/UCS with quartz content and quartz to feldspar ratio (Figure 4.2.2b and 4.2.3b), as shown for SAG in Section 3.3.6.1. In particular, samples with feldspar megacrysts showed the greatest increase in tensile strength with increased quartz to feldspar ratio (QFR) (Figure 4.2.4). If the ratio between UCS and tensile strength is taken as an indication of brittle potential, then this suggests that the presence of feldspar megacrysts (resulting in a lower QFR) would increase the brittle failure potential of granitic rock.
Figure 4.2.2: Quartz content versus UCS, Tensile and PLT Strength data (a) and ratios between Tensile, UCS and PLT (b) (data from Tuğrul and Zarif (1999)) showing increasing strength trends with increasing quartz content.
Figure 4.2.3: Quartz feldspar ratio versus UCS, Tensile and PLT Strength data (a) and ratios between Tensile, UCS and PLT (b) (data from Tuğrul and Zarif (1999)) showing increasing strength trends with increasing quartz to feldspar ratio.
Figure 4.2.4: Quartz feldspar ratio versus tensile strength (data from Tuğrul and Zarif (1999)), separated into groupings according to grain size distribution. This also shows an increasing strength trend with increasing quartz to feldspar ratio, but the increase for rocks with very coarse grains (megacrysts) increases at a faster rate than homogeneous or rock with evenly distributed range in grain size (seriate).

UCS perpendicular to fabric of foliated granites and gneisses shows that rocks have a weak tendency to increase with both increased quartz and feldspar content, and a very weak tendency to increase with quartz to feldspar ratio (Figures 4.2.5 and 4.2.6) (Göransson, Persson and Wahlgren, 2004). The same analysis for less foliated rock is inconclusive (Figures 4.2.5 and 4.2.6). If the rocks are separated into groups based on their grain size distribution (homogeneous and non-homogeneous) the same exercise shows that increased feldspar and quartz content, and QFR, tends to increase UCS, both parallel and perpendicular to foliation (Figures 4.2.7 and 4.2.8). The limited amount of data makes it difficult to separate the effects of grain size distribution and foliation from mineralogy, and therefore, to draw firm conclusions. The data can, however, be interpreted to support the findings of Lajtai (1998) and Tuğrul and Zarif (1999) in some cases, and at the very least is not robust enough to contest their findings.
Figure 4.2.5: Mineral content versus UCS, separated according to intensity of foliation (data from Göransson et al. (2004)); showing weak positive relationships between quartz (a) and feldspar (b) content of well foliated rocks and UCS, both parallel and perpendicular; poor, or no relationship found between quartz (a) and feldspar (b) content of poorly foliated rocks and parallel or perpendicular UCS.
A careful review of the data presented in Přikryl (2001) shows a weak relationship between increased feldspar content and strength, and increased QFR, and quartz content, leading to a decrease in strength. A closer inspection of the data revealed that a large percentage of the plagioclase grains were sericitised. The plagioclase content was modified for seritisation, and rocks were grouped according to their degree of seritisation. By taking this into account, the feldspar and QFR relationships become inconclusive, and the relationship between quartz and strength becomes weakly positive, thereby having little or slight positive impact on the conclusions of Lajtai (1998) and Tuğrul and Zarif (1999).

Figure 4.2.6: Quartz feldspar ratio versus UCS, separated according to intensity of foliation (data from Göransson et al. (2004)); showing a weak relationship between QFR and UCS strength for rocks with strong foliation, but little or no relationship for rocks with weak foliation.
Figure 4.2.7: Relationships between mineral content and UCS for foliated rocks with homogeneous grain size distribution only (data from Göransson et al. (2004)); showing increasing UCS with increased quartz content in the perpendicular loading direction, and the same trend for feldspar content and UCS in the perpendicular loading direction only.

Figure 4.2.8: Quartz to feldspar ratio versus UCS for all rocks and those with homogeneous grain size distribution (data from Göransson et al. (2004)); showing increasing trend between QFR and UCS in both loading directions.
The importance of grain boundaries and cleavage planes, considered as planes of weakness, has been investigated (i.e. (Kranz, 1983). In Westerly granite crack densities increase at a greater rate in feldspars than in quartz with increasing stress and cracks tended to follow feldspar and biotite to a greater extent than would be predicted by their modal percentage contributions, demonstrating the importance of cleavage cracking (Moore and Lockner, 1995).

The importance of mica has been demonstrated by several authors (Li, 2001; Li et al, 2003; Tapponier and Brace, 1976; Wong, 1982). Mica will deform by basal glide and kinking, unless the basal planes are oriented perpendicular to the applied load, and their ability to accommodate strain depends on their concentration, orientation to the loading direction, and degree and characteristics of alignment (Shea and Kronenberg, 1993). Isolated micas will tend to initiate fractures in the stronger phases, in particular when they are oriented at a low to moderate angle to the applied load. They also found that quartz and feldspar deform solely by microcracking, unless they constitute isolated inclusions in a mica matrix, in which case they remain relatively undeformed.

A few studies have been conducted in mafic rocks. Biotite and plagioclase have a higher microfracture density than hornblende and pyroxene in deformed gabbro, but gabbro has less overall damage than similarly deformed Westerly granite (Wong and Biegel, 1985). Cracking in amphibolite is concentrated in the more compliant phases (plagioclase and possibly quartz), which are present as a very small percentage, and stress anisotropies and strain concentrations are induced by minerals with high physical property contrasts compared to the matrix (Hacker and Christie, 1990). The impact of alteration of minerals is not commonly addressed, although hydrothermal alteration of plagioclase may interfere with long cleavage crack development due to interruption of cracks by replacement minerals (Moore and Lockner, 1995).

4.2.2.2 Synthesis of Yield Behaviour and Fracture Characterisation Studies

By taking these observations into account and considering the elastic and mechanical properties of minerals (Tables 4.2.4 and 4.2.5) the following relationships can be deduced:

1. Low stiffness, high anisotropy biotite does not propagate fractures well, but is a locus of fracture initiation, which then propagates through neighbouring minerals.

2. Moderately stiff quartz may be subject to tensile stress due to the elastic stiffness difference between it and the higher stiffness feldspar, leading to the initiation of tensile microcracks in the quartz.
3. Isotropic minerals (i.e. quartz) will propagate fractures parallel to the loading direction, while the cleavage of anisotropic minerals (i.e. feldspars, hornblendes and pyroxenes, and micas) will slow, arrest or change the direction of fractures propagating at a high angle to the cleavage or will facilitate intragranular crack propagation when favourably oriented.

4. The presence of feldspar megacrysts or feldspars with a preferred orientation will tend to favour the yielding of the feldspar grains and decrease strength.

5. High stiffness minerals may shatter first, leaving behind a low stiffness grain that may localise stress in adjacent grains (Diederichs, 1999), much the way micas and pores do.

6. The presence of a very stiff or very compliant mineral, even at low percentages, can have a disproportionate impact on the rock yield strength and yield behaviour.

7. Grain boundaries are important initiators of fractures, but are not considered in a mineralogy classification.

8. Hornblende and pyroxene are less prone to fracture than are plagioclase and biotite in gabbros.

9. The hydrothermal alteration of minerals will impact the behaviour of fractures through altered minerals, and this should be taken into account in the determination of mineral composition.

Differences in the elastic properties of minerals may contribute to cracking and more compliant minerals will lead to local stress heterogeneity and concentrations of elastic strain (Hacker and Christie, 1990; Tapponier and Brace, 1976), which can lead to tensile stresses in a compressive stress field (Diederichs, 2003). The tensile stress promotes tensile cracking, which dominates the local damage process in confined and unconfined compressive test simulations (Diederichs, 2003).

As a rock is strained, minerals with higher stiffness will attract more stress than more compliant minerals. If they also have high yield strength, they will require a high level of stress before they begin to fracture. If they have low yield strength, in particular if they have favourably oriented cleavage planes, then they will not be able to maintain the stress and will fracture. Compliant minerals will often begin straining early, dissipating the applied stress as they deform. This ductile behaviour may impose higher stress on adjacent stiff minerals. This can lead to tortuosity of the stress flow and zones of increased compressive stress as well as zones of tensile stress (Diederichs, 1999), thereby promoting tensile crack formation. By this logic, a monomineralic rock would have lab yield strengths approaching the lab yield strengths of the minerals themselves, with yield strength reduction arising from grain boundary cracks and healed microcracks, which are commonly induced between, and propagate through, minerals.
polymineralic rock the stiffness of different materials will result in distributions of stress and strain that depend on mineral types and orientation with respect to the applied stress. The presence of grains with contrasting elastic and mechanical properties will impact the rock yield strength and yield behaviour. For example, stiff, but moderately strong minerals may attract high stress and fail first, thus contributing to brittle yield of the rock.

The increased percentage of weak, compliant platy minerals will stem the propagation of unstable fractures (Diederichs, 2004) due to their ability to plastically deform by kinking or slip on cleavage planes (Li et al, 2003; Tapponier and Brace, 1976), and separation of cleavage planes and grain boundaries (Li et al, 2003) to accommodate strain. Their lower yield strength will decrease the rock’s yield strength at excavation boundaries while the inability to propagate unstable fractures will lower the ratio between lab and excavation boundary yield strength. If the percentage of weak, compliant minerals is great enough that the stronger, stiffer minerals no longer touch, the stiffer minerals will no longer contribute to the strength of the rock (Passchier and Trouw, 1996). In this case the deformation mechanism will change from brittle to ductile, in the engineering sense, where shear failure will dominate over tensile failure (Shea and Kronenberg, 1993).

The group of rocks used by Diederichs et al. (2004) gives an indication of the impact of the different mineral compositions on the reduction of lab yield strength. For example, quartzite is shown to have a high excavation boundary versus lab yield strength ratio, compared to the granites (Figure 4.2.9). This suggests that a nearly monomineralic rock has less strength reduction than a polymineralic rock. The marble falls in the middle of the list of rocks, despite its being a monomineralic rock. This would suggest that among monomineralic rocks, those composed of anisotropic minerals (such as calcite, feldspar, mica, etc.) have a greater strength reduction than those composed solely of quartz, which is isotropic.
Based on physical, elastic and lab strength mineral property information and the previous discussions, the minerals are grouped into two groups:

- \( F_{MM} \): Quartz, feldspar, amphibole, pyroxene and calcite make up the stiff, relatively isotropic major minerals group.

- \( F_{MA} \): Biotite, muscovite and chlorite make up the platy minerals group and garnet, pyrite and magnetite make up the very stiff group of the accessory minerals group.

The impacts of the two mineral groups are then combined to obtain the \( F_M \) factor. The results of the classification of the major, \( F_{MM} \), and accessory, \( F_{MA} \), mineral groups are numbered 1 to 3 and 1 to 4, respectively. Figure 4.2.10 demonstrates how \( F_M \) is obtained by combining the values. The designations for high, medium and low (H, M and L) for the mineralogy factor refer to the impact on strength reduction that particular combinations of minerals will have.
4.2.2.3.1 Grouping of Major Minerals – Determination of $F_{MM}$

The major minerals group is considered to be composed of minerals that act as fracture receptors. Feldspar and calcite are grouped together due to their cleavage, similar moderate stiffness and moderate anisotropy. Amphibole and pyroxene are grouped together due to their moderate anisotropy, similar crystallographic structure and high stiffness. Anisotropy in amphiboles and feldspars is “large”: up to 0.7 for amphiboles and from 0.5-0.8 for feldspars (Hacker and Christie, 1990), but compared to the platy minerals, this anisotropy is considered moderate here. Quartz and olivine are grouped together due to their isotropy and lack of cleavage, despite differences in stiffness. In nature these two minerals are very rarely found together so the stiffness difference is considered unimportant.

Although both olivine and quartz are grouped together, their geological associations are quite different, as is the impact of their presence in rocks. As such, two ternary plots demonstrate the impact, in high, medium and low, on excavation boundary strength reduction of different mineral compositions, one for mafic-felsic rocks (Figure 4.2.11) and one for mafic-ultramafic rocks (Figure 4.2.12). The graphs are based on relative mineral proportions, once the minerals of the platy/very stiff group (Group 2) are removed. Each graph has a zone of geologically unlikely compositions, which are not addressed in this research. The implication of this graph is that the divisions between high, medium and low impact on excavation boundary strength reduction can be reduced to quartz/olivine content, as a ratio of total mineral content. This greatly simplifies the classification scheme, making it more amenable to engineering application.
Figure 4.2.11: Ternary plot of stiff, nearly isotropic minerals group (felsic) showing the zones with mineral combinations that have been designated as having low, and high, impact on strength reduction at the excavation boundary.

Figure 4.2.12: Ternary plot of stiff, nearly isotropic minerals group (mafic) showing the zones of mineral combination designated as having low, medium and high impact on excavation boundary strength reduction.
Table 4.2.2: High, medium and low designation for \( F_{MS} \) according to quartz or olivine content

<table>
<thead>
<tr>
<th>Designation</th>
<th>Felsic-Mafic (Quartz content)</th>
<th>Mafic-Ultramafic (Olivine content)</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>High (1)</strong></td>
<td>20-60%</td>
<td>~</td>
</tr>
<tr>
<td><strong>Medium (2)</strong></td>
<td>60-80%</td>
<td>20-80%</td>
</tr>
<tr>
<td><strong>Low (3)</strong></td>
<td>80 &lt; quartz &lt; 20%</td>
<td>80 &lt; olivine &lt; 20%</td>
</tr>
</tbody>
</table>

The major minerals are grouped in zones of high, medium and low and are assigned \( F_{MS} \) values from 1 to 3, respectively (Table 4.2.2). Rocks that are nearly monomineralic, or have a mafic composition with low quartz/olivine content will tend to have excavation boundary strength approaching lab yield strength due to little perturbation of the stress field for the monomineralic case and composition of high strength/elastic properties minerals for the mafic case. Rocks with a combination of quartz/olivine and other minerals will have a moderate impact on strength reduction, up to 60% quartz and throughout the entire spectrum for olivine content due to the impact of differing elastic properties on the stress field, without a considerable impact from feldspars, which are highly anisotropic. Rocks with quartz content between 20 and 60% will have high strength reduction due to the high stress perturbation and the presence of feldspars which, while cracking late in the stress history, will fail suddenly (Li et al, 2003) and contribute to early (in terms of stress) excavation boundary brittle yielding, compared to lab yielding.

### 4.2.3.2 Grouping of Accessory Minerals – Determination of \( F_{MA} \)

The minor minerals group is considered to be composed of minerals that act as fracture initiators. Micas and platy minerals are capable of making up a large portion of a rock (i.e mica schist, chlorite schist), but above a certain percentage the effect of the platy minerals on the rock strength changes from fracture initiation (leading to brittle yield) to ductile yield accommodated by slip along the basal planes. Büchi (1988) found that above 30% mica content, rocks tend to behave in a similar fashion regardless of increasing mica content. Diederichs et al. (2004) have put this limit at 20% mica content while Shea and Kronenberg (1993) have found that brittle yield occurs in rocks with less than 21% mica, and that ductile behaviour can begin to occur in rocks with greater than 28% mica content.

For the platy minerals, the anisotropy increases from muscovite to biotite and chlorite, while the stiffness similarly decreases. Garnet, magnetite and pyrite, while commonly found in crystalline rocks, do not often constitute a large percentage of the rock, although minerals making up a low percentage of the rock can still influence the strength of brittle rock if their elastic modulus is vastly different from the major minerals making up the rock (Tapponier and Brace, 1976). They have, therefore, been included in this classification because of their conspicuously
high stiffness. To differentiate between the impact of the minerals themselves, they have been grouped according to their anisotropy and stiffness into: chlorite and biotite, having highest impact; muscovite having moderate impact; and garnet, pyrite and magnetite having lowest impact (Figure 4.2.13). Only the relative mineralogy is used, and in essence only noting which mineral makes up the majority of this group is necessary.

The cut-off for effect of micas on excavation boundary strength reduction is selected as 20%, above which the rocks do not fit into this model and should be considered as ductile rocks with different yield characteristics than those being described in this research. If the mica content is less than 20%, the rock is classified into a high (2 to 10%) and low (11 to 20%) impact of the mica minerals on strength reduction (Table 4.2.3). The two classifications are combined as shown in Figure 4.2.14, giving the category for FMP.

![Figure 4.2.13: Ternary plot of platy minerals and very stiff minerals showing the relative mineral combinations related to low, medium and high impact on strength reduction at the excavation boundary](image)

<table>
<thead>
<tr>
<th>Designation</th>
<th>Group 2 content (total)</th>
<th>Greatest proportion (mineral)</th>
</tr>
</thead>
<tbody>
<tr>
<td>High</td>
<td>10-20%</td>
<td>Biotite/chlorite</td>
</tr>
<tr>
<td>Medium</td>
<td>-</td>
<td>Muscovite</td>
</tr>
<tr>
<td>Low</td>
<td>2-10%</td>
<td>Garnet/pyrite/magnetite</td>
</tr>
</tbody>
</table>

Table 4.2.3: High, medium and low designations for $F_{MA}$ according to initiator content in the rock and which mineral makes up the highest percentage of this group.
Figure 4.2.14: FMA designations, from 1 to 4, as a combination between the percentage of initiators in the rock and the relative % of the greatest fraction of initiators

### 4.2.2.4 Mineral Properties

Three different types of mineral properties were considered in the determination of similar mineral groupings for the different minerals found in the rocks under consideration in this research. The first type is the physical properties, which are most often used by geologists for mineral identification. The second type is elastic properties, also used by geologists, mostly for rheology studies. The third type is lab strength properties, used nearly exclusively by geological engineers and researchers for studies concerning rock strength and fracture toughness. The following discussion addresses the applicability of each of these property types to creating functional mineral groupings for geomechanical characterisation.

#### 4.2.2.4.1 Mineral Physical Properties

The properties in Table 4.2.4 are of value in mineral identification and for geological purposes but they do not lend themselves to aid in grouping similar minerals since there are several combinations of properties and no clear separation of different mineral groups.

Fracture toughness is a key mineral property that has been investigated for its impact on rock strength and yield behaviour (Atkinson and Avdis, 1980; Broz, Cook and Whitney, 2006; Tromans and Meech, 2002) but it does not necessarily correspond to hardness (Broz, Cook and Whitney, 2006), precluding the use of hardness (for example Moh’s hardness scale) to group minerals for their impact on strength and yield behaviour.
Table 4.2.4: Simplified physical characteristics of common rock forming minerals in felsic granitoids and metamorphic rocks (from Nesse (1991))

<table>
<thead>
<tr>
<th>Mineral</th>
<th>Moh’s Hardness</th>
<th>Cleavage/Parting/Fracture</th>
<th>Tenacity</th>
<th>Alteration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Quartz</td>
<td>7</td>
<td>None/none/conchoidal</td>
<td>Brittle</td>
<td>None</td>
</tr>
<tr>
<td>Potassium Feldspar</td>
<td>6-6.5</td>
<td>1 perfect; 1 good; 90°</td>
<td>Brittle</td>
<td>Sericite</td>
</tr>
<tr>
<td>Plagioclase Feldspar</td>
<td>6-6.5</td>
<td>1 perfect; 1 good; 93-94°</td>
<td>Brittle</td>
<td>Sericite</td>
</tr>
<tr>
<td>Amphibole</td>
<td>5-6</td>
<td>2 good; 56, 124°/1-2 partings</td>
<td>Brittle</td>
<td>Chlorite</td>
</tr>
<tr>
<td>Pyroxene</td>
<td>5-6</td>
<td>2 good; 88°/2 partings</td>
<td>Brittle</td>
<td>Amphibole</td>
</tr>
<tr>
<td>Calcite</td>
<td>3</td>
<td>1 perfect; 74°</td>
<td>Ductile</td>
<td>Aragonite</td>
</tr>
<tr>
<td>Biotite</td>
<td>2.5-3</td>
<td>1 perfect</td>
<td>Flexible/elastic/ductile</td>
<td>Chlorite</td>
</tr>
<tr>
<td>Muscovite</td>
<td>2.5-3</td>
<td>1 perfect</td>
<td>Flexible/elastic/ductile</td>
<td></td>
</tr>
<tr>
<td>Chlorite</td>
<td>2-3</td>
<td>1 perfect</td>
<td>Flexible/ductile</td>
<td>Clay</td>
</tr>
<tr>
<td>Garnet</td>
<td>6.5-7.5</td>
<td>Non/non/conchoidal-uneven</td>
<td>Brittle</td>
<td></td>
</tr>
<tr>
<td>Magnetite</td>
<td>5.5-6</td>
<td>None/1 parting</td>
<td>Brittle</td>
<td>Hematite, limonite, goethite</td>
</tr>
<tr>
<td>Olivine</td>
<td>6.5-7</td>
<td>2 poor</td>
<td>Brittle</td>
<td>Serpentine, mixture of clays and silicates</td>
</tr>
<tr>
<td>Pyrite</td>
<td>6-6.5</td>
<td>4 poor/none/conchoidal</td>
<td>Brittle</td>
<td></td>
</tr>
</tbody>
</table>

4.2.2.4.2 Determination of Mineral Elastic Properties

Table 4.2.5 contains the physical and elastic properties for the minerals in Table 4.2.4. Most of the values were obtained, as is, from the cited publications listed below, and some were calculated from elastic constants of the crystals or from compressibility measurements.

The values contained in Birch (1966) are initial compressibility values designated by ‘a’. This value can be assumed to equal the compressibility of the mineral at zero confining pressure, designated as $\beta_0$, which can be used to calculate the bulk modulus at zero confining pressure, $K_0$ (Birch, 1966):

$$K_0 = \frac{1}{\beta_0}$$

4.2.1

The bulk modulus was then used to calculate the elastic (Young’s) modulus, $E$, using Poisson’s ratio, $\nu$, according to the following relationship:

$$E = 3K(1 - 2\nu)$$

4.2.2
Table 4.2.5: Average physical and elastic properties of rock forming minerals in Table 4.2.4

<table>
<thead>
<tr>
<th>Mineral</th>
<th>Density g/cm³</th>
<th>Young’s Modulus GPa</th>
<th>Bulk Modulus GPa</th>
<th>Poisson’s Ratio</th>
<th>Source</th>
</tr>
</thead>
<tbody>
<tr>
<td>Quartz</td>
<td>2.65</td>
<td>55.74</td>
<td>-</td>
<td>0.19</td>
<td>1</td>
</tr>
<tr>
<td></td>
<td></td>
<td>90</td>
<td>-</td>
<td>-</td>
<td>3</td>
</tr>
<tr>
<td></td>
<td></td>
<td>67.8</td>
<td>-</td>
<td>-</td>
<td>5</td>
</tr>
<tr>
<td></td>
<td></td>
<td>66.7</td>
<td>36</td>
<td>-</td>
<td>2</td>
</tr>
<tr>
<td>Alkali Feldspar</td>
<td>2.55</td>
<td>71</td>
<td>47-52</td>
<td>0.28</td>
<td>1</td>
</tr>
<tr>
<td></td>
<td></td>
<td>62.7-68.5c</td>
<td>67.8</td>
<td>-</td>
<td>2</td>
</tr>
<tr>
<td>Plagioclase Feldspar</td>
<td>2.57</td>
<td>75.1</td>
<td>-</td>
<td>0.27</td>
<td>1</td>
</tr>
<tr>
<td></td>
<td></td>
<td>95.6</td>
<td>-</td>
<td>-</td>
<td>6</td>
</tr>
<tr>
<td></td>
<td></td>
<td>68.3-78.6c</td>
<td>49.5-57</td>
<td>-</td>
<td>2</td>
</tr>
<tr>
<td>Amphibole</td>
<td>3.05</td>
<td>106.53</td>
<td>83-91</td>
<td>0.26</td>
<td>1</td>
</tr>
<tr>
<td></td>
<td></td>
<td>119.5-131.7c</td>
<td>-</td>
<td>-</td>
<td>2</td>
</tr>
<tr>
<td>Pyroxene</td>
<td>3.32</td>
<td>149</td>
<td>93</td>
<td>-</td>
<td>1</td>
</tr>
<tr>
<td></td>
<td></td>
<td>133c</td>
<td>-</td>
<td>-</td>
<td>2</td>
</tr>
<tr>
<td></td>
<td></td>
<td>-</td>
<td>-</td>
<td>0.26</td>
<td>7</td>
</tr>
<tr>
<td>Calcite</td>
<td>2.71</td>
<td>85</td>
<td>73</td>
<td>-</td>
<td>1</td>
</tr>
<tr>
<td></td>
<td></td>
<td>94c</td>
<td>-</td>
<td>-</td>
<td>2</td>
</tr>
<tr>
<td></td>
<td></td>
<td>-</td>
<td>-</td>
<td>0.285</td>
<td>4</td>
</tr>
<tr>
<td>Biotite</td>
<td>2.75</td>
<td>20</td>
<td>35.2</td>
<td>-</td>
<td>1</td>
</tr>
<tr>
<td></td>
<td></td>
<td>51.7EH</td>
<td>-</td>
<td>-</td>
<td>2</td>
</tr>
<tr>
<td></td>
<td></td>
<td>52.7GH</td>
<td>-</td>
<td>-</td>
<td>2</td>
</tr>
<tr>
<td>Muscovite</td>
<td>2.79</td>
<td>80.4</td>
<td>52.6</td>
<td>0.25</td>
<td>1</td>
</tr>
<tr>
<td></td>
<td></td>
<td>82.1c</td>
<td>-</td>
<td>0.26</td>
<td>2</td>
</tr>
<tr>
<td></td>
<td></td>
<td>53EH</td>
<td>-</td>
<td>-</td>
<td>2</td>
</tr>
<tr>
<td></td>
<td></td>
<td>53GH</td>
<td>-</td>
<td>-</td>
<td>2</td>
</tr>
<tr>
<td>Chlorite</td>
<td>2.78</td>
<td>60.9</td>
<td>-</td>
<td>-</td>
<td>1</td>
</tr>
<tr>
<td>Garnet average</td>
<td>4.22</td>
<td>239</td>
<td>166c</td>
<td>0.26</td>
<td>1</td>
</tr>
<tr>
<td>Magnetite</td>
<td>4.11</td>
<td>103.2</td>
<td>57c</td>
<td>0.2</td>
<td>1</td>
</tr>
<tr>
<td>Olivine</td>
<td>3.32</td>
<td>217</td>
<td>139c</td>
<td>0.24</td>
<td>1</td>
</tr>
<tr>
<td>Pyrite</td>
<td>4.85</td>
<td>-</td>
<td>146</td>
<td>0.17</td>
<td>1</td>
</tr>
<tr>
<td></td>
<td></td>
<td>289c</td>
<td>-</td>
<td>-</td>
<td>2</td>
</tr>
</tbody>
</table>

* assume plagioclase and potassium feldspar have similar laboratory strength properties

* calculated based on relationship between elastic modulus, bulk modulus and Poisson’s ratio

EH, CH values for elastic modulus and rigidity modulus, respectively, in direction perpendicular to the basal plane of mica

1 Lama and Vutukuri (1978)
2 Birch (1966) – initial compressibility
3 Li et al. (2003)
4 Holbrook (2000)
5 Carmichael (1982)
6 Angel et al. (1988)
7 Kumazawa (1969)

The elastic constant values from Birch (1966) were used for mica since no compressibility values are provided, except for phlogopite mica (K=43 GPa, E=51.2 GPa). The elastic constants are denoted by simplified tensor notation as follows: $c_{ik}$ and $s_{ik}$, where the $c_{ik}$ is the stiffness and $s_{ik}$ is the compliance, either of which can be used to determine the elastic, shear and bulk moduli (Hearmon, 1956). The suffixes i and k are simplifications of tensor suffixes, for example: $c_{pqrs} = c_{1123} = c_{ik} = c_{14}$, where i = 1 when pq = 11 and k = 4 when rs = 23 (a complete...
explanation can be found in Hearmon, 1956). It has been shown (as discussed in McNeil and
Grimsditch, 1993) that micas and chlorite, as well as clays, have monoclinic symmetry, however,
when using elastic constants to calculate the bulk modulus, mica is assumed to have hexagonal
symmetry (Alexandrov and Ryzhova, 1961; Birch, 1966). The Voigt or Reuss methods for
determining aggregate elastic moduli (Hearmon, 1956) use the elastic constants from hexagonal
symmetry (although some of the constants, i.e. c_{22} (s_{22}) and c_{55} (s_{55}) can be input separately
without using the hexagonal assumption that c_{11}=c_{22}, etc.) because some of the elastic constants
that are non-zero in monoclinic symmetry are not used in either of these methods. Although there
is some anisotropy within the silica sheets, which have covalent bonding, the greatest anisotropy
in micas arises across the sheets due to the mostly ionic bonding between them (McNeil and
Grimsditch, 1993). The anisotropy between the basal planes in the normal elastic constants in
muscovite is approximately 3, while in the shear direction it is approximately 5 (McNeil and
Grimsditch, 1993) and in biotite and chlorite the normal anisotropy is nearly 3.5 and the shear
anisotropy is nearly 15 (Alexandrov and Ryzhova, 1961).

The Voigt method, which assumes uniform strain and non-uniform stress, overestimates
the aggregate elastic moduli while the Ruess method, which assumes uniform stress and non-
uniform strain, underestimates them (Hill, 1952). Since neither is quite true, and the measured
values fall between the Voigt and Reuss values, their average should be used for aggregates (Hill,
1952). The Voigt and Reuss values are similar for materials with higher symmetry (i.e. cubic
Hill, 1952) but the spread becomes greater at lower symmetries. Here, neither the Voigt nor
Reuss methods can be used on their own to determine the elastic moduli of mica using elastic
constants of the crystals.

A further complication arises due to the high anisotropy of the elastic constants. An
aggregate average can be used to compare micas to the other minerals, but their elastic properties
in the weakest direction (perpendicular to the basal planes) gives an indication of the behaviour of
these minerals along their weakness plane. These values are, therefore, also included based on
the relationships:

\[ E_i = \frac{1}{s_{ik}} \quad \text{for } i = k \quad \text{and} \quad G = \frac{1}{s_{ik}} \quad \text{for } i = k \quad \text{and} \quad i, k > 3 \]  

For the purposes of this study, s_{33} will be used to determine the elastic modulus in the
direction perpendicular to the basal planes of the mica. Similarly, s_{66} will be used to determine
the rigidity modulus in the same direction. The anisotropy within the basal plane is assumed to
be negligible compared to the anisotropy perpendicular to it.
4.2.2.4.3 Standard Laboratory Strength Properties

The laboratory strength properties in Table 4.2.6 are also used to adjust the mineral groupings in the model by grouping minerals with similar strength properties. Values for tensile strength and fracture toughness are not available for several minerals. While a number of values for fracture toughness are available, they comprise both laboratory and theoretical values and cannot be compared to each other effectively. The UCS values have some variability, but they demonstrate strength trends that can be used with the elastic properties to group minerals together.

Table 4.2.6: Average laboratory strength properties of rock forming minerals in Table 4.2.4

<table>
<thead>
<tr>
<th>Mineral</th>
<th>UCS</th>
<th>Tensile</th>
<th>UCS/σ</th>
<th>KIC MNm^{-3/2}</th>
<th>Source</th>
</tr>
</thead>
<tbody>
<tr>
<td>Quartz</td>
<td>222.6</td>
<td>33.78</td>
<td>6.59</td>
<td>-</td>
<td>1</td>
</tr>
<tr>
<td></td>
<td>200</td>
<td>10</td>
<td>20</td>
<td>0.3825</td>
<td>3</td>
</tr>
<tr>
<td></td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>1.6</td>
<td>2</td>
</tr>
<tr>
<td>Alkali Feldspar</td>
<td>180</td>
<td>36</td>
<td>5</td>
<td>-</td>
<td>3</td>
</tr>
<tr>
<td></td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>0.393</td>
<td>2</td>
</tr>
<tr>
<td>Plagioclase Feldspar</td>
<td>180*</td>
<td>36*</td>
<td>5*</td>
<td>-</td>
<td>3</td>
</tr>
<tr>
<td></td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>0.393*</td>
<td>2</td>
</tr>
<tr>
<td>Amphibole</td>
<td>297</td>
<td>18.25</td>
<td>16.27</td>
<td>-</td>
<td>1</td>
</tr>
<tr>
<td>Pyroxene</td>
<td>150-180</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>1</td>
</tr>
<tr>
<td>Calcite</td>
<td>120-150</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>1</td>
</tr>
<tr>
<td></td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>0.187</td>
<td>2</td>
</tr>
<tr>
<td></td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>0.39</td>
<td>5</td>
</tr>
<tr>
<td>Biotite</td>
<td>150</td>
<td>-</td>
<td>-</td>
<td>0.393</td>
<td>1</td>
</tr>
<tr>
<td></td>
<td>120</td>
<td>12</td>
<td>10</td>
<td>0.393</td>
<td>3</td>
</tr>
<tr>
<td>Muscovite</td>
<td>112</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>1</td>
</tr>
<tr>
<td>Chlorite</td>
<td>112</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>1</td>
</tr>
<tr>
<td>Garnet_average</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>1.272</td>
<td>4</td>
</tr>
<tr>
<td>Magnetite</td>
<td>200</td>
<td>23.2</td>
<td>8.62</td>
<td>-</td>
<td>1</td>
</tr>
<tr>
<td></td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>1.683</td>
<td>4</td>
</tr>
<tr>
<td>Olivine</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>0.96</td>
<td>2</td>
</tr>
<tr>
<td>Pyrite</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>1.297</td>
<td>4</td>
</tr>
</tbody>
</table>

* assume plagioclase and potassium feldspar have similar laboratory strength properties
1 Lama and Vutukuri (1978)
2 Atkinson and Avdis (1980)
3 Li et al. (2003)
4 Tromans and Meech (2002); theoretical values for perfect crystals
5 Broz et al. (2006)
4.2.3 Grain Size and Distribution Classification – $F_G$

4.2.3.1 Geomechanical Characterisation of Grain Size and Grain Size Distribution

In nearly all rock types grain size is a significant parameter for strength (Riedmüller and Schubert, 2001), although it is not currently regularly taken into consideration in rock yield behaviour prediction. Grain size is also an important factor in in-situ versus lab strength reduction, as will be discussed. There exist several geological methods for quantifying and classifying grain size as it is an important aspect of determining rock origin and history. Some of the methods used by geologists can be applied to rock characterisation for engineering purposes by considering how changes in grain size and distribution will affect the internal stress condition, fracture behaviour and, ultimately, the excavation boundary relative to the lab strength.

The characterisation factor $F_G$ was developed to account for grain size and grain size distribution. As shown in Figure 4.2.1 $F_G$ is a combination of two geological characteristic factors $F_{GP}$ and $F_{GD}$, representing petrological grain size and grain size distribution, respectively. Petrological grain size refers to the grain size represented by exterior grain boundaries of individual minerals and is a choice between three grain size categories: small (<0.5mm), medium (0.5-5mm) and large (>5mm). Grain size distribution is a characterisation of type of distribution of the entire series of grain sizes, categorised into two groups: isotropic or bimodal with <10% clasts, and seriate (graded) or bimodal with >10% clasts. As will be discussed in Section 4.2.3.2, some minerals can have tectonically induced grain size modification, sometimes resulting in reduced effective grain size. The term effective grain size is used to describe the presence of obstacles to fracture propagation similar to grain boundaries and is accounted for by the tectonic grain size factor, $F_{GT}$, which may be used to modify the $F_{GP}$ factor. The combination of $F_{GP}$ ($\pm F_{GT}$) and $F_{GD}$ leads to three possible categories for $F_G$ related to its impact on rock yield behaviour at the excavation boundary: low, medium and high.

4.2.3.2 Grain Size Impact on Rock Yield strength

The general consensus is that yield strength tends to decrease as grain size increases (Howarth and Rowlands, 1987; Tuğrul and Zarif, 1999; Singh et al., 2001). Grain size has been shown to be the most important microstructural factor influencing granite yield strength in mineralogically similar rock types (Prikryl, 2001), especially strong rocks with moderate
anisotropy. Tensile strength varies to a greater extent with grain size than does compressive strength (Onodera and Asoka-Kumara, 1980), perhaps due to the greater percentage of microfractures in coarser mineral grains. These would be more sensitive to direct tension, and would impact the strength much more than in compression where tension only occurs when isolated tensile zones are created.

Relationships between grain size and yield strength have been investigated in material science. A common relationship, the Hall-Petch relationship, describes the relationship between grain size ($d$) and yield strength ($\tau_y$) of metals, as follows (Illston, Dinwoodie and Smith, 1979):

$$\tau_y = \tau_i + K \cdot d^{-n}$$

For body centered cubic metals, $n = \frac{1}{2}$, but it may vary for metals with other crystal structures. The stress at which dislocations begin to move, is represented by $\tau_i$, which is related to the material in question. $K$ is a material constant (Illston, Dinwoodie and Smith, 1979). This relationship is often applied to rocks and other polycrystalline aggregates with considerable success by Brace (1961) for quartzite; McCombs et al. (1974) for ceramics; and Nasser et al. (2005) for original microfracture length and grain size.

Of particular interest is how grain size affects the yield behaviour of rocks. For marbles and limestones, the grain size effect on strength follows a Hall-Petch relationship during failure at room temperature, but attempts to predict yield strength based on the relationship between grain size and microfracture size significantly underestimate the yield strength, suggesting that spatial local stress field heterogeneity should scale with the grain size, and could be more fundamentally responsible, rather than the size of original flaws, for the effect of grain size on rock strength (Fredrich et al., 1990). This indicates that increased grain size will lead both to increased microfracture length and increased spatial stress field heterogeneity, both of which contribute to lower rock strength.

Coarser grained natural amphibolite fails in a brittle fashion under a variety of temperature conditions, but finer grained synthetic amphibolite reaches the brittle-ductile transition and thus fails by ductile means (Hacker and Christie, 1990), suggesting that coarser grain size may increase the potential for in-situ strength reduction in amphibolites. Grain size was a dominating factor for tunnel face stability over several hundreds of metres in gneisses and granites excavated by TBM, indicating a relationship between coarse grained rock and increased depth of failure in the tunnel face, compared to fine grained rock (Weh and Bertholet, 2005). This increased depth of failure may be due to decreased rock strength or to higher susceptibility to in-situ strength reduction (and thus spalling potential).
4.2.3.3 Microfracture Impact on Rock Yield strength

The nature and behaviour of microfractures is complex and investigations into their effect on yield strength and stress-induced fracture behaviour are numerous. Early work by Brace (1961) concluded that the Griffith crack length is of the same order of magnitude as the grain size, which has been supported by other authors (i.e. Nur and Simmons, 1970; Onodera and Asoka-Kumara, 1980). Strength was found to vary with the inverse square root of marble grain size, and to have a critical initial microfracture density above which the strength becomes relatively insensitive to increased microfracture density (Wong et al., 1996). Rock strength decreases with increased grain size through a process by which proportionately longer cracks propagate along lengthier weakness planes, which are more likely to coalesce at lower stresses (Eberhardt et al., 1999). Nasseri et al. (2002) found a good correlation between microfracture length and rock fracture toughness. Originally existing microcracks play a more important role in plastic strain and eventual failure of mylonites than the formation of new cracks in quartz and feldspar layers (Brosch et al, 2000) and intragranular cracks constitute three-quarters of the sum of macrofracture lengths, with grain boundary fractures accounting for only one quarter (Moore and Lockner, 1995). In Westerly granite 35% of new fractures are generated along grain boundaries and 65% through quartz and feldspar (Wawersik and Brace, 1971).

It is evident that there is both a relationship between microfractures and strength, and between microfractures and grain size. The two combined may account for a portion of the relationship between grain size and strength, however, the goal of this research is to develop a method by which easily obtained geological characteristics can be translated into valuable information for engineering applications, not necessarily unravelling the intricate relationships between all discernable characteristics and rock strength and yield behaviour. For this reason, microfractures are not considered and their contribution to in-situ strength reduction is taken into account indirectly when characterising the grain size and grain size distribution of a rock.

4.2.3.4 Model Development

The grain size and distribution classification takes into account the maximum, minimum and median grain size as well as the distribution of the grain sizes (uniform, seriate or bimodal), and the proportion of clasts and matrix. The grain size, $F_{GP}$, and grain size distribution, $F_{GD}$, each contribute to $F_G$, as shown in Figure 4.2.15, with $F_{GT}$ acting to reduce $F_{GP}$ by one grain size category if it reaches a threshold value of 0.5.
4.2.3.4.1 Grain Size Petrological– Determination of $F_{GP}$

For this research, the median, rather than average, petrological grain size is used to reduce the effect of outliers, such as very large plagioclase porphyroclasts. The grain size can be determined through thin section or hand sample analysis, although the latter is much less precise. For geological descriptive purposes the grain size can be classified by the scheme in Table 4.2.6, which is based on the International Union of Geological Sciences (IUGS) and Wentworth grade scale classification for sedimentary rocks, and provides a uniform scheme with similar class divisions for sedimentary, volcanic, igneous and metamorphic rocks. While this scale is valuable for geological rock description, its level of detail is greater than is actually necessary for engineering application. For this reason, the grain size is broken into three groups, as shown in Figure 4.2.15.

Table 4.2.6: Grain size classification according to the International Union of Geological Sciences (IUGS) and Wentworth grade scale (after Gillespie and Styles (1999))

<table>
<thead>
<tr>
<th>Grain size</th>
<th>Class Name</th>
</tr>
</thead>
<tbody>
<tr>
<td>&gt; 16mm</td>
<td>Very coarse grained</td>
</tr>
<tr>
<td>2 – 16mm</td>
<td>Coarse grained</td>
</tr>
<tr>
<td>0.25 – 2mm</td>
<td>Medium grained</td>
</tr>
<tr>
<td>0.032 – 0.25mm</td>
<td>Fine grained</td>
</tr>
<tr>
<td>0.004 – 0.032mm</td>
<td>Very fine grained</td>
</tr>
<tr>
<td>&lt;0.004mm (4μm)</td>
<td>Cryptocrystalline</td>
</tr>
</tbody>
</table>
4.2.3.4.2 Grain Size Tectonic – Determination of $F_{GT}$

In geology, specifically, and materials science, in general, research into plastic deformation processes at the grain scale constitutes an important aspect of determining the history of a rock and the strength of the material, respectively. The natural approach to investigating rock strength characteristics based on rock properties includes the exploration of the impact of plastic deformation. Knowledge of the different plastic deformation processes that rocks undergo in nature and the manner in which they can be identified and interpreted constitutes one portion of the approach. The second portion is an understanding of the effects of different deformation histories on metal’s yield behaviour and ultimate strength. Combining them into a model for in-situ strength prediction forms the final portion of the approach, and has proven to be complex.

The different aspects of geological deformation can be found in Appendix B.2 and a discussion of the metals and materials science analogy can be found in various publications, including (Hacker and Christie, 1990; Illston et al., 1979; Nesse, 1991; Passchier and Trouw, 1996) and is summarised in Appendix D.1 (Villeneuve et al, 2005).

In rocks, plastic deformation and recovery processes occur separately only in rare occasions (Nicolas and Poirier, 1976), so the features of both deformation and recovery will be present at different magnitudes depending on the tectonic history and mineralogy. While materials science advances can be applied to deformation of rock in the ductile regime, the analogy does not apply as directly in the brittle domain. In this domain the movement of dislocation and their interactions is slower than the rate at which stresses build up and lead to fracture. What can be drawn from deformation features of minerals is the development of undulatory extinction, subgrain boundaries and grain size reduction. Grain size reduction leads to the creation of new grain boundaries and undulatory extinction and subgrain boundary formation create boundaries approaching the characteristics of grain boundaries (Nicolas and Poirier, 1976) (Figure 4.2.16), which create obstacles to the propagation of fractures, thereby reducing the potential for in-situ strength reduction.
Figure 4.2.16: Schematic demonstrating examples of features resulting from ductile deformation of quartz, and how they are used to modify grain size when such features are present.

The $F_{GT}$ factor is related to the percentage of grains, within the entire population of grains of that mineral type, with deformation features that give evidence for strengthening. Quartz and calcite display strengthening and weakening features (Table 4.2.7) that are visible under transmitted light microscopy and can therefore be used to determine the percentage of strengthening or weakening of the mineral grains due to tectonic strains. The rest of the minerals identified in Table 4.2.7 either have no deformation mechanisms, besides twinning, or have unknown mechanisms. Twinning does not contribute to mineral strength (Nicolas and Poirier, 1976) and is therefore not used to determine $F_{GT}$. This leaves quartz and calcite, which are major minerals found in felsic crystalline rocks, pelitic gneisses, meta-sedimentary rocks such as marble, and sedimentary rocks, and often form subgrains in response to ductile deformation (Liu, Walter and Weber, 2002; Nicolas and Poirier, 1976).

The percentage value for quartz and/or calcite (if present) can be used alone, if only one of these minerals is present, or as a weighted average of both percentages, if both minerals are present. This classification is, therefore, a refinement of the grain size of the visible grains.
Table 4.2.7: Deformation and recovery of minerals from Table 4.2.3 (Nicolas and Poirier, 1976)

<table>
<thead>
<tr>
<th>Mineral</th>
<th>Dominant deformation/ recovery mechanisms</th>
</tr>
</thead>
<tbody>
<tr>
<td>Quartz</td>
<td>Increasing deformation corresponding to: deformation lamellae, subgrain boundary formation, subgrain rotation</td>
</tr>
<tr>
<td></td>
<td>Increasing recovery corresponding to: subgrain rotation, grain boundary migration, grain boundary area reduction</td>
</tr>
<tr>
<td>Alkali Feldspar</td>
<td>N/A</td>
</tr>
<tr>
<td>Plagioclase Feldspar</td>
<td>Pericline and Albite mechanical twinning</td>
</tr>
<tr>
<td>Amphibole</td>
<td>Mechanical twinning: difficult to activate</td>
</tr>
<tr>
<td>Pyroxene</td>
<td>Mechanical twinning</td>
</tr>
<tr>
<td>Calcite</td>
<td>Increasing deformation corresponding to: mechanical twinning and slip leading to subgrain formation</td>
</tr>
<tr>
<td></td>
<td>Increasing recovery corresponding to: grain boundary migration, grain boundary area reduction</td>
</tr>
<tr>
<td>Biotite</td>
<td>Bending, kinking; recrystallisation very difficult</td>
</tr>
<tr>
<td>Muscovite</td>
<td>N/A</td>
</tr>
<tr>
<td>Chlorite</td>
<td>N/A</td>
</tr>
<tr>
<td>Garnet average</td>
<td>N/A</td>
</tr>
<tr>
<td>Magnetite</td>
<td>N/A</td>
</tr>
<tr>
<td>Olivine</td>
<td>Some recrystallisation at grain boundaries</td>
</tr>
<tr>
<td>Pyrite</td>
<td>N/A</td>
</tr>
</tbody>
</table>

To account for the modal percentage of quartz or calcite in the whole rock, the $F_{GT}$ factor is defined as:

$$F_{GT} = B_Q Q + B_C C$$

where $B_Q$ and $B_C$ are the percentage of undulose extinction or grain boundaries present in quartz and calcite, respectively, and $Q$ and $C$ are the modal percentages of calcite and quartz, respectively. When $F_{GT}$ is greater than 0.5, the $F_{GP}$ value will drop by one grain size. For example, the grains shown in Figure 4.2.16 would be classified as medium sized grains when viewed at hand sample scale (in which case only the exterior boundary is discernable). Under the microscope, however, the undulose extinction (left) and subgrain boundaries (right) become visible. In this example, albeit with only two grains, the $F_Q$ value would be 100%.

4.2.3.4.3 Grain Size Distribution – Determination of $F_{GD}$

While considerable previous work has been conducted on the analysis of grain size on rock strength, very little has been done regarding grain size distribution. It has been shown, however, that non-uniform grain size distribution contributes to stress flow tortuosity and, hence, increases the likelihood of tensile stress zones occurring within a compressive stress field (Diederichs, 2003). Additionally, as previously shown, increasing grain size leads to increased likelihood of unstable crack propagation due to the increased length of single transgranular cracks forming in response to applied stress (Eberhardt et al., 1999; Nasseri et al., 2005; Brace, 1961).
The grain size distribution can be qualitatively classified as uniform, seriate or bimodal.

- uniform describes a narrow range of grain sizes,
- seriate describes a wide range of grain sizes with good representation from grains in a series of sizes (similar to graded for granular material), and
- bimodal describes two distinct sets of grain sizes.

The bimodal distribution class is subdivided into three categories related to percentage of larger clasts within the finer matrix, corresponding to classification of deformed rocks (Robertson, 1999) as follows: > 50% clasts, 10 to 50% clasts, < 10% clasts. This is important because the strength of a polymineralic rock does not change linearly with the percentage of hard or soft minerals since few strong minerals will rotate in a matrix of weak minerals and only contribute to strength if they are in contact with each other (Passchier and Trouw, 1996). The same can be said of porphyroclasts in a grain size reduced, weaker matrix, where the strength of the clasts only becomes important when they can support stress by pressing on each other. By this logic, bimodal grain size distribution can either be interpreted as leading to a rock with properties arising solely from the matrix (<10% clasts), strength arising mainly from the matrix (10 to 50% clasts) or strength arising from a combination of matrix and clasts (>50% clasts). In Figure 4.2.15 the rocks with bimodal distribution with <10% clasts are grouped with the uniform rocks, while those with >10% clasts are combined with rocks having seriate grain size distribution.

Increased grain size and non-uniform distribution contribute to lowered in-situ strength and increased lab versus in-situ strength ratio. Decreased percentage of clasts in bimodally distributed grain size contributes to decreased in-situ strength and decreased lab versus in-situ strength ratio. The presence of feldspar megacrysts in a rock may reduce the tensile strength and lead to more brittle behaviour (Tuğrul and Zarif, 1999/2). If there are fewer than 10% clasts, however, they are unlikely to impact the rock behaviour. Although rocks with very fine grained micas would exhibit lower yield strength than large grained micas, because of their ductile behaviour in this configuration, their lab yield strength would approach in-situ strength, and they therefore fall within the regime of reduced lab versus in-situ strength ratio represented by fine grained rock.
4.2.4 Fabric Classification – $F_A$

4.2.4.1 Geomechanical Characterisation of Foliation

The proposed foliation factor, $F_A$, is an excavation boundary strength reduction and anisotropy intensification factor (Figure 4.2.1). It relates foliation type, $F_{AF}$: mineral preferred orientation, gneissic layering, schistosity or cleavage, to microlithon size, $F_{AD}$, whose scale divisions vary according to selected foliation type. The selection of foliation type is based on size of minerals defining foliation, continuity of foliation and mineral segregation. The selection of microlithon size is simply based on the scale of microlithons between identifiable foliation planes. The resulting $F_A$ factor is a designation of low, medium or high impact on fracture potential, the designation of which depends on the scale of the problem under consideration and is denoted separately according to scale; in this case: TBM cutter scale, $F_{A1}$, or tunnel face scale $F_{A2}$. The process by which $F_{A1}$ and $F_{A2}$ are obtained is identical, and is referred to herein generically as $F_A$; only the outcome is different depending on the scale of interest.

4.2.4.1.1 Definition of Fabric and Foliation

Fabric is distinguished by the alignment of platy or elongate minerals within a plane or the segregation of mineral types into bands while maintaining cohesion along the planes (Marshak and Mitra, 1998). The degree of fabric penetration, from isotropic to schistose to gneissic and mylonitic, is typically classified as it relates to increasing metamorphic grade. Joints, fractures, faults and partings along cleavage are not considered foliation by this definition because they lack cohesion.

Foliation can be broken into two groups: spaced and continuous (Passchier and Trouw, 1996). Spaced contains two layers: the cleavage layer with aligned grains or compositional layer and the microlithon layer of unaligned grains between the cleavage layers. Continuous cleavage contains a non-layered alignment of platy or elongate minerals. These could be undeformed platy minerals, such as mica and amphibole, or deformed minerals, such as quartz, feldspar and calcite. Foliation can also be divided into three groups: cleavage, schistosity and gneissic layering (Marshak and Mitra, 1998). Cleavage and schistosity are characterised by domains that have been affected by the cleavage and microlithons that have not (Marshak and Mitra, 1998). The distinction between cleavage and schistosity is the grain size of the platy minerals according to the following: cleavage grains are not visible to the naked eye, while grains defining schistosity are. Gneissic layering is described as mineral composition layering of fine to medium grains,
which may or may not be oriented, that can range in thickness from microscopic to local scale (Davis and Reynolds, 1996).

4.2.4.1.2 Importance of Fabric to Rock Yield Strength and Behaviour

For foliated rocks, like phyllites, foliation properties and orientation are key parameters for predicting ground behaviour in tunnel construction (Riedmüller and Schubert, 2001). Failure patterns in schistose rocks can be very different, depending on the existence of continuous weakness planes, and the presence of weakness planes leads to different yield mechanisms in unconfined versus confined compression tests. In addition, the UCS is highest at very high and very low angle between the loading direction and the weakness planes and a minimum at an angle between 30 and 45° (Nasseri et al., 1997). Although mica content can be related to rock yield strength and yield behaviour, both are more directly affected by the initial contiguity of mica in weakness planes, along which fractures may form. Mica grain, and aggregate, characteristics play a more dominant role in controlling rock yield strength than do variations in the non-phylosilicates (Shea and Kronenberg, 1993). Weakness planes representing aligned, contiguous mica clusters most directly influence yield strength and yield behaviour, and, even at low mica contents, the orientation of individual mica grains at shallow angles to the loading direction can also affect strength (Shea and Kronenberg, 1993). In order to include the effect of fabric on rock yield behaviour a visual/quantitative model was developed to combine both the tools available from tectonics and metamorphic geology as well as engineering needs for quantification.

4.2.4.2 Synthesis of Fabric and Texture Classification Studies

For rocks with continuous cleavage (as per the definition by Passchier and Trouw, 1996) the FM could be determined first by looking at a variety of rock types with fabric and performing an alignment test (i.e. Li et al, 2003; Wong, 1982 for microcracks). Characterising fabric by measuring orientation of defects and grain elongation (i.e. Willard and McWilliams, 1969) would not be capable of properly characterising rocks with spaced foliation since it only addresses preferred mineral alignment, not spacing between foliation planes and microlithons.

The availability of high powered computers and image analysis software has led to the development of complex characterisation schemes for texture or fabric and their comparison to rock yield strength or yield characteristics (i.e. Launeau and Robin, 1996; Prikryl, 2001) some of which is rigorous for research purposes, but is too demanding for engineering application (i.e. (Bathurst and Rothenburg, 1990; Lisle, 1985). The dimensionless texture coefficient (TC)
Howarth and Rowlands (1987) measures grain circularity, elongation, orientation and grain packing, but is only demonstrated for isotropic rocks and does not take into account the anisotropy related to segregation of minerals into foliation planes and the impact of microlithon thickness on resulting block size.

Some techniques still require considerable data analysis to be performed on thin section images by hand, which can sometimes also be performed by computer programs (i.e. Brosch et al., 2000). The foliation index (FI) represents the length to width ratios of individual mineral grains normalised by the modal percentage of minerals, and is highly dependent on the orientation in which the thin section is cut and the actual three dimensional shape of the mineral grains. It is difficult to apply to very fine grained rocks since it requires X-ray diffraction for application to these rock types (Noble Tsidzi, 1990). Measurements of mica cluster and quartzo-feldspathic bridge dimensions compared to rock yield strength and yield behaviour suggests a strong relationship between fabric and fracture behaviour, as well as UCS strength (Shea and Kronenberg, 1993). Foliation can be quantified by measurements of numbers of grain boundaries intersecting transects in thin section or mineral grain perimeters by scanning electron microscope (SEM) (Åkesson et al, 2003). These methods were only demonstrated on granites, which are not intensely foliated, and therefore the application of these methods to this study is not justified. In addition, the difficulty in obtaining the information necessary to characterise the rock makes it unsuitable for engineering purposes.

While several techniques for characterising fabric have been developed, no single one was found to completely fulfill the goals of this research. Neither TC nor FI are able to represent visually classified characteristics for mylonitic gneiss with porphyroclasts (Brosch et al, 2000). The majority of the techniques involve considerable effort due to their dependence on sophisticated computer image analysis techniques or laborious hand analysis techniques, more suitable to research than to rigorous engineering application. A simple, scale independent fabric index for mica minerals for two fabric considerations was developed by Büchi (1988): the direction of mica minerals in a homogeneous distribution and the continuity of the foliation as defined by micas in a heterogeneous distribution. Each consideration is broken into four classes whose indices are added to obtain the mica fabric index. Its simplicity makes this method suitable for engineering application; however, it is rather subjective and does not directly consider microlithon spacing.

Interlocking texture seems to play a role in isotropic rocks (Howarth and Rowlands, 1987), however, it is not sufficient for anisotropic rocks where continuous weakness planes play a very important role in both yield strength and yield characteristics of rocks (Nasseri et al., 1997;
Büchi, 1988; Shea and Kronenberg, 1993). Linear mica continuities, length and width of mica clusters and dimensions of quartzo-feldspathic bridges can be related to rock fracture mode where increased quartzo-feldspathic bridge dimension combined with decreased mica cluster dimensions was shown to lead to more brittle yield and higher yield strength; decreased linear mica continuity was also shown to lead to more brittle fracture and higher yield strength (Shea and Kronenberg, 1993). Åkesson et al. (2003) found that mica content alone could not explain differences in resistance to rock fracture in aggregate production, but that mineral grain perimeter measurements could, and that the foliation index was useful to predict particle shape resulting from aggregate production, suggesting that both should be used to predict aggregate shape.

4.2.4.3 Model Development

The evidence presented in the literature and work by previous authors led to the development of a foliation characterisation model that will be applicable to engineering purposes and will describe the variations in fabric that affect brittle fracture. The relationship between a hypothetical FA (and thus strength and anisotropy) and two different fabric characterisation schemes can be seen in Figure 4.2.17. The qualitative description method on the right is analogous to descriptions of foliated metamorphic rocks, and leads to a non-unique relationship between this FA and the fabric. By using a quantitative measure of fabric, such as the one on the left, a linear or direct relationship can be drawn, which is simpler to use.

![Figure 4.2.17](image)

Figure 4.2.17: Schematic showing relationship of FA factor to two alternative methods of classifying fabric, where the image on the left lends itself most to engineering application, whereas the image on the right is based on genetic geological classification of fabric.
The proposed characterisation model consists of a chart (Figure 4.2.18) in which the vertical axis describes characteristics related to foliation plane characteristics and thickness, $F_{AF}$, while the horizontal axis describes characteristics related to microlithon description and thickness, $F_{AD}$. Individual $F_A$ values are associated with each combination of $F_{AF}$ and $F_{AD}$, represented in Figure 4.2.18 as individual squares. The $F_A$ factor is used to determine the effect on fracture promotion for rocks with a single or several foliations, as discussed in Section 4.3.

### 4.2.4.3.1 Foliation Type Selection – $F_{AF}$ and Associated Microlithon Spacing – $F_{AD}$

Three foliation groups, cleavage, schistosity and gneissic layering, are used as the basis for categories describing the characteristics and thickness of the foliation. A fourth category, mineral preferred orientation, describes mineral alignment that is not contiguous. Each group has foliation characteristics that are important indicators of geological history, are easily described by geologists and have implications for mechanical behaviour. The four $F_{AF}$ categories represent a range in size of grains defining: i) the foliation from subgrain size (not visible to the naked eye) described by cleavage, ii) grain size (visible to the naked eye) described by schistosity, iii) several grain sizes related to gneissic layering, and finally iv) the mineral preferred orientation, which has variable grain size. The two end members of the axis are defined at continuous and discontinuous related to the description of microlithon characteristic and thickness, $F_{AD}$. This axis has a variable
scale depending on the type of foliation being described, and can only be selected once the foliation type, \( F_{AF} \), on the vertical axis has been determined.

The classification of disjunctive cleavage \( F_{AD} \) is based on domain spacing and morphology (Marshak and Mitra, 1998). Continuous cleavage (for example slaty cleavage) has a pervasive alignment of platy minerals in which discrete microlithons are not visible at the hand specimen scale (Marshak and Mitra, 1998), while spaced cleavage is separated by visible microlithons. Three ranges of cleavage spacing, \( F_{AD} \), were selected for the model: less than 0.5 cm, 0.5 to 5 cm and greater than 5 cm. These are related to chip thickness under TBM cutters and the intensity of the cleavage, where the first range is very strong, the second is strong to moderate and the third is moderate to weak (Figure 4.2.19). Examples of rocks classified with each of the three microlithon ranges are shown in Figures 4.2.20 to 4.2.22.

![Figure 4.2.19: Schematic showing cleavage intensity divisions (after Marshak and Mitra (1998)); divisions used for the microlithon thickness in the cleavage classification](image-url)
Figure 4.2.20: Photomicrograph of cleavage with spacing <0.5mm, as used in this classification (GA_a102)

Figure 4.2.21: Photomicrograph of cleavage with spacing 0.5 - 5mm, as used in this classification (GA_095)
Classification of schistosity type and scale, \( F_{AD} \), is based on morphology and broken into three groups: domainal, continuous type 1 and type 2 (Marshak and Mitra, 1998). Domainal schistosity contains domains of platy minerals that anastomose around domains of non-platy minerals, such as quartz or feldspar (Figure 4.2.23), and are linearly contiguous across a thin section. Continuous type-1 schistosity has preferred alignment of coarse platy minerals with the possibility of microlithons only visible under the microscope (Figure 4.2.24). Continuous type-2 schistosity is defined by preferred orientation of any mineral within the rock in which no domains are visible at any scale (Figure 4.2.25).
Figure 4.2.23: Photomicrograph of domainal schistosity, as used in this classification (GA_a012)

Figure 4.2.24: Photomicrograph of Type 1 schistosity, as used in this classification (GA_c127)
Gneissic banding is categorised by layer thickness, FAD, which can be considered to be similar to cleavage but on a larger scale from less than 3 cm, 3-10 cm and greater than 10 cm. Mineral preferred orientation (Figure 4.2.26) is only domainal, but the platy domains are not contiguous across a thin section. In this material, fractures must eventually pass through microlithons, which can slow their progression. This foliation category is, therefore, broken into two FAD groups: less than 1cm spacing and greater than 1cm spacing. This is related to the likelihood of fractures being induced by the oriented minerals located close enough to each other to interact and form macro-fractures.
4.2.4.3.2 Classification for Cutter – $F_{A1}$ and Tunnel Face Scale – $F_{A2}$

The $F_A$ factor is related directly to chipping at the cutter scale or fracture promotion at the tunnel face scale. $F_A$ values are quoted according to their application to estimating rock behaviour under the cutter, $F_{A1}$, or at the tunnel face scale, $F_{A2}$. Decreasing the $F_A$ factor will lead to decreased excavation boundary strength parallel to the fabric and increased anisotropy between the directions parallel and perpendicular to the fabric. The chart in Figure 4.2.27 contains the impact of each of the foliation categories on excavation boundary strength reduction at the cutter scale and tunnel face scale, designated as ++, + and o for high, medium and low, respectively. Figure 4.2.28 shows how different foliated rock types are to be classified by the scheme presented in Figure 4.2.27, first by selecting the fabric type, $F_{AF}$, and then by selecting the appropriate microlithon scale and characteristic, $F_{AD}$. 
Figure 4.2.27: Schematic of FA classification scheme showing impact of each foliation category on cutter and tunnel face scale excavation boundary strength reduction. $F_{A1}$ and $F_{A2}$ represent cutter and face scale effects, respectively; magnitude of effect represented by: ++ for high, + for moderate and o for low.
Figure 4.2.28: Schematic of FA classification scheme with interpretations of where various typical foliated rock types plot on the chart, shown as hatched zones.

### 4.2.5 Fracture Potential Determination

As discussed in Section 4.2.1, the three F Factors, $F_M$, $F_G$, and $F_A$, developed in the previous sections are combined to obtain a measure of the magnitude by which lab yield strength can be reduced, $F_{SS}$, and lead to the fracture potential, $F_{Fr}$, at the excavation boundary. The $F_M$ factor describes the impact that different combinations of mineralogy will have on the yield
behaviour. The different minerals under consideration are grouped according to their elastic stiffness and lab strength properties. Minerals with similar combinations of strength and stiffness were grouped together as major minerals, $F_{MM}$, and accessory minerals, $F_{MA}$. The $F_G$ factor considers the impact of both grain size, $F_{GP}$ and $F_{GT}$, and its distribution, $F_{GD}$, which, in combination, lead to differences in yield behaviour, where some promote fracture growth and propagation, while others blunt it. The $F_A$ factor relates to the intensity of the foliation present in tectonically deformed rock. The fabric categories, $F_{AF}$, take into account the size of platy particles defining foliation, and are categorised as $F_{AD}$ according to the size of microlithons, the non-platy mineral’s domains. Each of the $F_M$, $F_G$ and $F_A$ require inputs from interpretation of the mineralogical and textural features of the rock. The most efficient and reliable way of interpreting these features is with a combination of thin section and hand sample analysis. The $F_A$ factor can be improved by additional information found in hand samples and at the tunnel scale, either in rock at the surface above the tunnel alignment or in nearby valleys.

### 4.2.5.1 Spall Sensitivity – $F_{SS}$

The spalling sensitivity, $F_{SS}$, reflects the magnitude by which laboratory strength values can overestimate the actual strength of rocks at excavation boundaries. This value provides a lower bound estimate of this apparent strength reduction, and is related to the ability of fractures to first initiate and then propagate through rocks under the particular confining stress regime found at excavation boundaries, as described in Section 4.1. Isotropic spalling sensitivity is an isotropic description of yield strength and behaviour at the excavation boundary, and its impact does not depend on the direction of loading. Factors $F_M$ and $F_G$ are considered isotropic factors, and thus have the same application for both isotropic and anisotropic crystalline rocks. The $F_A$ can be applied to rocks with single or multiple fabrics, with the caveat that as the number of fabrics increases, the anisotropy will decrease. This will become important in Chapter 5 when the $F$ Factors are compared to TBM performance. Their product describes the spall sensitivity, $F_{SS}$, as follows:

$$F_{SS} = F_M \times F_G \times F_A$$

### 4.2.5.2 Fracture Potential – $F_{FI}$

The fracture potential, $F_{FI}$, is the predicted, lower bound yield strength under the confining stress regime found at excavation boundaries and is an estimate of the likelihood of
sudden rock fracture at excavation boundaries. It is based on the spalling potential, $F_{SS}$, and relates this value to the expected behaviour at the excavation boundary. In this research, the fracture potential is of particular importance at two different scales: the TBM cutter scale, in which case chipping behaviour is important, and the tunnel face scale, in which case face instability is important. The fracture potential value is computed as follows:

$$F_{FI} = F_{SS} \times UCS \times \left[ F_M \times F_G \times F_A \right] \times \text{labstrength}$$

using the $F_{SS}$ value and standard lab strength values, such as UCS. Since the fracture potential takes into account the impact of planes of weakness on excavation boundary strength and rock behaviour, the orientation of the planes of weakness to the tunnel and the induced stress are important, requiring additional information before the $F_{FI}$ can be used. For rocks with several overprinting fabrics, whose behaviour approaches isotropic behaviour, the orientation of the fabrics is not as critical and the $F_{FI}$ can be used directly.
4.3 Classification of the Geomechanical Characterisation Scheme for Chipping Performance

4.3.1 Geomechanical Characterisation for Cutter and Tunnel Face Scales

The TBM performance data analysis methodology developed in Section 3.2 was used to determine the geological dependence of TBM chipping performance and tunnel face stability. As discussed in Section 3.2, each arises from different, but related spalling-type failure processes. The sensitivity of different rock types to these processes can be anticipated by examining their geomechanical characteristics, as discussed in Section 4.2.

The work by Diederichs et al. (2004) demonstrates that weighted geological factors can be used to estimate in-situ excavation boundary strength in conjunction with laboratory test values, such as UCS. The relative importance of the F-Factors described in Section 4.2 to spalling sensitivity was determined using a rigorous literature review in which testing data or descriptions of fracture behaviour were interpreted. A large suite of thin sections (approximately 250) from the set of samples collected in the Southern Aar granite (see Section 2.2.2) were characterised according to the system described in Section 4.2. The unweighted characterisation results are presented in Appendix B.4.

The results from the characterisation were processed using a Receiver/Response Operating Characteristic curve (Sackett et al, 1991) to determine the two following attributes:

- The relative impact of each F-Factor on chipping performance
- The thresholds at which a change in behaviour is expected

These were combined to create a classification system by which the F-Factors from the geomechanical characterisation scheme are used to determine the chipping performance of different rock types.

4.3.2 Quantitative Geomechanical Characterisation

4.3.2.1 Introduction

In order to use the F-Factors to anticipate the behaviour of particular rock types during underground excavation the framework in which they are organised, the Geomechanical
Characterisation scheme, must be quantified. This will allow the user to anticipate if a particular set of geological characteristics is likely to lead to spalling type failure, and make the appropriate interpretations for failure behaviour during underground excavation. A first pass at quantification and threshold determination based on test results and interpretations available in the literature was presented in Section 4.2. A more rigorous method was employed to quantify the relative impacts of each F-Factor on chipping performance, which is a spalling-type failure. The raw F-Factor data (uncategorised) were collected using thin section analysis according to Appendix D.2. The TBM performance for the strokes related to each sample location were used to characterise the rock as chipping or non-chipping according to the methodology developed in Section 3.2 for the start up dataset from the Southern Aar granite Altkristallin. The two were combined to test each F-Factor for its ability to predict chipping performance and determine the threshold at which each F-Factor indicates changes in chipping performance.

4.3.2.2 ROC Curves for Chipping Performance

The data collected by thin section analysis represent the real geological conditions encountered by the TBMs. The information from the Southern Aar granite was used to create ROC curves for each F-Factor. Since the data are real and highly variable, subsets were used to determine relationships.

4.3.2.2.1 Samples Without Fabric

Samples without fabric were used to determine the relationship between chipping performance and mineralogy, grain size and grain size distribution. The ROC curves for accessory mineral content were calculated first since a clear distinction between rocks that are chipping and non-chipping was possible at 8% accessory mineral content. All mineralogy, grain size and grain size distribution ROC curves were calculated for the entire dataset without fabric, as well as the dataset without fabric with accessory mineral content less than 8%.

The data relating to the major minerals were investigated in terms of quartz content, feldspar content and quartz to feldspar ratio. The amphibole/pyroxene content was too low in these rocks to effectively investigate the impact of these minerals. Since the amphibole/pyroxene content was not investigated, the quartz to feldspar ratio was an effective way to investigate the interrelationship in the major minerals along the amphibole/pyroxene content = 0% line from Figure 4.2.10.
The area under the ROC curve in Figure 4.3.1 is 0.72, which demonstrates that mica content has a fair accuracy for predicting chipping performance. The threshold of 8% mica separates samples with only chipping designations from samples with mixed designations. For this reason, subsequent ROC curves are performed on samples with less than 8% mica content. The graph in Figure 4.3.2 is based on the dominant type of accessory mineral, for the entire no-fabric dataset. The area under the curve is 0.58, and is not acceptable for designating chipping performance, however, an examination of the data resulting from the ROC analysis indicates that biotite/chlorite is a predictor for chipping performance, while muscovite is not. This is possibly due to the texture of most of the muscovite present in the sample dataset, arising from alteration of feldspar, and thus without the platy crystal shapes with which the biotite and chlorite exhibit.

The area under the ROC curve in Figure 4.3.3 is 0.72, which demonstrates that mica content has a fair accuracy for predicting chipping performance. The threshold of 4% mica gives the best results for most true positive predictions and fewest false negative predictions.

Figure 4.3.1: ROC curve for mica content (all samples)
Figure 4.3.2: ROC curve for accessory mineral type (all samples)

Figure 4.3.3: ROC curve for mica content, for subset with mica content less than 8%
The area under the ROC curve for quartz content, feldspar content and quartz to feldspar ratio (Figures 4.3.4 – 4.3.6) are 0.78, 0.79 and 0.77, respectively, which demonstrates that quartz to feldspar ratio has a fair accuracy for predicting chipping performance. The threshold of 0.6 for the quartz to feldspar ratio gives the best results for most true positive predictions and fewest false negative predictions.

Figure 4.3.4: ROC curve for quartz content, for subset with mica content less than 8%
The ROC curve in Figure 4.3.7 has an area of 0.51, which demonstrates that the effectiveness of grain size data in this configuration for chipping performance prediction is inadequate, and must be configured to highlight the impact of grain size on the chipping process. If the grain sizes are divided into ranges and weighted according to their impact on chipping, then
the area under the corresponding ROC curve is 0.62 (Figure 4.3.8), a considerable, although poor, improvement, demonstrating that grain size does impact the chipping process. The ranges selected for analysis are based on an interpretation of Figure 4.3.7 and the selection of two thresholds corresponding to changes from good chipping above 1mm, mixed good and poor chipping between 0.5 and 1mm, and poor chipping below 0.5mm.

The ROC curve in Figure 4.3.9 has an area of 0.68, which demonstrates that grain size distribution is a poor predictor for chipping performance. In particular, ‘isotropic’ and ‘bimodal’ are better predictors than is ‘seriate’. The grain size ranges were combined with the grain size distribution types and weightings were associated with the combinations according to their impact on chipping. The resulting ROC curve (Figure 4.3.10) has an area of 0.65, which is an improvement over simply using the grain size. These results suggest that combined grain size and grain size distribution categories are poor predictors for the chipping process, given this dataset.

![ROC curve for grain size, for subset with mica content less than 8%](image)

Figure 4.3.7: ROC curve for grain size, for subset with mica content less than 8%.
Figure 4.3.8: ROC curve for grain size, for subset with mica content less than 8%, separated into grain size ranges.

Figure 4.3.9: ROC curve for grain size distribution, for subset with mica content less than 8%.
4.3.2.2.2 Samples with Fabric

Samples with fabric were used to determine the relationship between chipping performance and fabric type and intensity. In order to isolate these relationships, samples with similar mineralogy, grain size and grain size distribution were grouped and analysed. The fabric in the Southern Aar granite was typically oriented perpendicular to the tunnel axis, and hence the tunnel face. This has been found to lead to improved excavation in terms of penetration rate (Figure 4.3.11, after Sanio, 1985), although in the Southern Aar granite, improved excavation sometimes led to face instability. The samples were, therefore, classified according to chipping performance as well as face instability.
Figure 4.3.11: Schematic of relationship between fabric orientation, and tunnel face and cutter orientation (after Sanio (1985)).

ROC curves were generated using undifferentiated sample data, where the only criterion was that the samples had fabric. These curves had areas between 0.51 and 0.58 for all of the above criteria, except for mica content, and including two additional criteria: fabric type and fabric intensity. The area for the ROC curve for mica content was 0.64, (Figure 4.3.12) in which the samples with mica content between 9-20% were found to be mixed chipping and non-chipping, while samples with less than 9% and samples with greater than 20% mica (22 samples, out of a total of 169) were found to have only chipping failure. The ROC curve area is not very high, due to the low percentage of samples with greater than 20% mica, but is an indication that 20% mica content is a more valid threshold for chipping performance of samples with fabric, than is the 8% threshold that was determined for rocks without fabric.

To investigate the impact of fabric type and intensity on chipping performance, values were assigned to $F_A$ and were weighted to obtain the highest ROC area. The resulting graph has an area of 0.69 (Figure 6.3.10), which suggests that fabric, weighted in this configuration is a poor-fair predictor of chipping performance.
Figure 4.3.12: ROC curve mica content for rocks with fabric

Figure 4.3.13: Chipping performance ROC curve for fabric factor $F_A$ selected for chipping prediction.
A similar approach was used to determine the impact of fabric on face instability in the Southern Aar granite, resulting in Figure 4.3.14. A modified F Factor specifically for face instability resulted in a ROC curve area of 0.61, suggesting that fabric is a poor predictor for face instability in this configuration. In addition to fabric type and intensity, face instability will depend on fabric orientation and in-situ stress condition in the tunnel face. In the Southern Aar granite, both the fabric orientation and in-situ tunnel face stress condition were favourable for face instability. The fabrics in this section of the Aar Massif are aligned with each other with little variability (as discussed in Section 2.3). For this reason, no data are available in this study to investigate the impact of fabric orientation on face instability. In addition, the in-situ stress condition in the tunnel face does not dramatically change over the 500m of tunnel from which data were collected, resulting in a lack of data regarding the impact of stress condition at the face on face instability in the Southern Aar granite.

![ROC Curve](image)

Figure 4.3.14: Face instability ROC curve for fabric factor $F_A$ selected for chipping prediction

### 4.3.3 Summary of Geomechanical Calibration for Chipping Performance

#### 4.3.3.1 F-Factor Thresholds

The thresholds for each F-Factor in the Geomechanical Characterisation scheme developed in Section 4.2 by literature review, were modified or confirmed by the thin section
The chipping performance is designated as low chipping, medium chipping and high chipping. Since insufficient data were available for rocks with amphibole or pyroxene, the major mineral characterisation is based only on the relative quartz and feldspar contents. Table 4.3.1 demonstrates the revised thresholds for chipping performance within the major mineral category, $F_{MM}$, based on relative quartz content.

Table 4.3.2 demonstrates the revised thresholds for accessory mineral content. The threshold for the accessory minerals group includes the clear threshold for high chipping performance in samples with accessory mineral content greater than 8%. In the subset with accessory mineral content less than 8%, the threshold at 4% again separates samples with high chipping performance from samples with low or mixed chipping performance (Table 4.3.2). Samples with biotite/chlorite as the greatest proportion of accessory mineral content were more likely to contribute to high chipping performance than samples with muscovite or garnet/pyrite/magnetite as the greatest proportion. The combined thresholds for the accessory mineralogy factor $F_{MA}$ are shown in Table 4.3.3 and the combined thresholds for the mineralogy factor $F_M$ are shown in Table 4.3.4.

Table 4.3.1: Chipping performance designation for major minerals according to relative quartz content ($F_{MM}$)

<table>
<thead>
<tr>
<th>Chipping Performance Designation</th>
<th>Felsic-Mafic (Quartz to Feldspar ratio)</th>
</tr>
</thead>
<tbody>
<tr>
<td>High</td>
<td>&lt;0.6</td>
</tr>
<tr>
<td>Medium</td>
<td>&gt;0.6</td>
</tr>
</tbody>
</table>

Table 4.3.2: Chipping performance designation for accessory minerals according to accessory mineral content and which mineral makes up the greatest proportion ($F_{MA}$)

<table>
<thead>
<tr>
<th>Chipping Performance Designation</th>
<th>Accessory Minerals Content</th>
<th>Greatest Proportion (mineral)</th>
</tr>
</thead>
<tbody>
<tr>
<td>High</td>
<td>&gt;8%</td>
<td>Biotite/chlorite</td>
</tr>
<tr>
<td>Medium</td>
<td>4-8%</td>
<td>Muscovite</td>
</tr>
<tr>
<td>High</td>
<td>2-4%</td>
<td></td>
</tr>
</tbody>
</table>

Table 4.3.3: Combined chipping performance designation for accessory minerals according to accessory mineral content and which mineral makes up the greatest proportion ($F_{MA}$)

<table>
<thead>
<tr>
<th>Chipping Performance Designation Content</th>
<th>Greatest Proportion Content</th>
<th>Biotite/ Chlorite</th>
<th>Muscovite</th>
</tr>
</thead>
<tbody>
<tr>
<td>&gt;8%</td>
<td>High</td>
<td>High-Medium</td>
<td></td>
</tr>
<tr>
<td>4-8%</td>
<td>Medium-High</td>
<td>Medium</td>
<td></td>
</tr>
<tr>
<td>&lt;4%</td>
<td>High</td>
<td>High-Medium</td>
<td></td>
</tr>
</tbody>
</table>
The thresholds for grain size are little changed from the original thresholds from Section 4.2, however, the grain size distribution thresholds are. The ROC curve for grain size (Figure 4.3.8) is inconclusive, however, a careful examination of the data analysis suggests that a grain size of 0.5mm provides a suitable threshold between chipping and non-chipping, while an examination of the dataset grouped according to sensitivity to chipping reveals that no samples with grain size greater than 1mm were characterised as non-chipping. This leads to the thresholds outlined in Table 4.3.5. The grain size distribution ‘seriate’ was found to provide inconclusive results for chipping performance designation. Isotropic and bimodal, however, provide a good indication for non-chipping and chipping, respectively. The combined thresholds for the grain size and grain size distribution factor FG are shown in Table 4.3.6.

Table 4.3.4: Combined chipping performance designation for mineralogy factor according to accessory and major mineral content (F_M)

<table>
<thead>
<tr>
<th>Chipping Performance Designation</th>
<th>F_MM</th>
<th>&lt;0.6</th>
<th>&gt;0.6</th>
</tr>
</thead>
<tbody>
<tr>
<td>High</td>
<td>High</td>
<td>High-Medium</td>
<td></td>
</tr>
<tr>
<td>Medium-High</td>
<td>High-Medium</td>
<td>Medium-High</td>
<td></td>
</tr>
<tr>
<td>Medium</td>
<td>Medium-High</td>
<td>Medium</td>
<td></td>
</tr>
</tbody>
</table>

Table 4.3.5: Chipping performance designation for grain size (F_GP) and grain size distribution (F_GD)

<table>
<thead>
<tr>
<th>Chipping Performance Designation</th>
<th>Grain Size F_GP</th>
<th>Grain Size Distribution F_GD</th>
</tr>
</thead>
<tbody>
<tr>
<td>High</td>
<td>&gt;1mm</td>
<td>Bimodal</td>
</tr>
<tr>
<td>Medium</td>
<td>0.5-1mm</td>
<td>Seriate</td>
</tr>
<tr>
<td>Low</td>
<td>&lt;0.5mm</td>
<td>Isotropic</td>
</tr>
</tbody>
</table>

Table 4.3.6: Chipping performance designation for combined grain size and grain size distribution (F_G)

<table>
<thead>
<tr>
<th>Chipping Performance Designation</th>
<th>F_GP</th>
<th>&gt;1mm</th>
<th>0.5-1mm</th>
<th>&lt;0.5mm</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bimodal</td>
<td>High</td>
<td>Medium-High</td>
<td>Medium</td>
<td></td>
</tr>
<tr>
<td>Seriate</td>
<td>Medium-High</td>
<td>Medium</td>
<td>Low-Medium</td>
<td></td>
</tr>
<tr>
<td>Isotropic</td>
<td>Medium</td>
<td>Low-Medium</td>
<td>Low</td>
<td></td>
</tr>
</tbody>
</table>
The thresholds for chipping performance with respect to fabric are based on both fabric type and spacing (or intensity), as described in Section 4.2. Only cleavage, schistosity and mineral preferred orientation are addressed here since none of the rock types encountered were classified as having gneissic fabric. The chipping performance designations for fabric type and intensity \((F_A)\) are shown in Table 4.3.7.

In addition to chipping performance, the tunnel face stability was also investigated with respect to the fabric factor. The thresholds for face instability do not mirror the thresholds for chipping performance (Table 4.3.8). The designations for high, medium and low face instability also vary, although in this case they vary roughly with intensity.

Table 4.3.7: Chipping performance designation for fabric type and intensity \((F_A)\)

<table>
<thead>
<tr>
<th>Chipping Performance Designation</th>
<th>(F_A)</th>
<th>Close spacing</th>
<th>Intermediate</th>
<th>Domainal</th>
</tr>
</thead>
<tbody>
<tr>
<td>(F_{AF})</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cleavage</td>
<td>High</td>
<td>Low</td>
<td>Medium</td>
<td></td>
</tr>
<tr>
<td>Schistosity</td>
<td>High</td>
<td>Low</td>
<td>Medium</td>
<td></td>
</tr>
<tr>
<td>Mineral Preferred Orientation</td>
<td>Low</td>
<td></td>
<td>Medium</td>
<td></td>
</tr>
</tbody>
</table>

Table 4.3.8: Face instability designation for fabric type and intensity \((F_A)\)

<table>
<thead>
<tr>
<th>Face Instability Designation</th>
<th>(F_A)</th>
<th>Close spacing</th>
<th>Intermediate</th>
<th>Domainal</th>
</tr>
</thead>
<tbody>
<tr>
<td>(F_{AF})</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cleavage</td>
<td>High</td>
<td>Low</td>
<td>Medium</td>
<td></td>
</tr>
<tr>
<td>Schistosity</td>
<td>High</td>
<td>High</td>
<td>Low</td>
<td></td>
</tr>
<tr>
<td>Mineral Preferred Orientation</td>
<td>Low</td>
<td></td>
<td>Medium</td>
<td></td>
</tr>
</tbody>
</table>

### 4.3.3.2 F-Factor Weightings

The best predictors for chipping performance are the relative quartz and feldspar contents, as well as the accessory mineral content. This is followed by a fair prediction for chipping performance based on grain size and grain size distribution, as well as fabric type and intensity. The relative ability of each factor to predict chipping performance is an indicator of the weighting the corresponding F Factors should be given. An examination of the work undertaken by Diederichs et al. (2004) gives a first indication of the weightings associated to rock types from
the literature to predict in-situ excavation boundary strength, which is likely similar to weightings necessary to predict chipping performance.

In the FSR classification (Diederichs et al., 2004) each factor other than the general rock type (F1) ranges from 0.75 to 0.85 or in the case of micas and other accessory minerals (F4) to 0.9. These factors are relatively similarly weighted. The work presented in Section 4.3.2.3 shows that each of the geomechanical characterisation scheme factors (F_M, F_G and F_A) has a different impact on the chipping performance, as interpreted based on the ROC curve areas. The quartz content (F_{MM}) relative to feldspar has the highest ROC curve area of 0.77, followed closely by mica content (F_{MA}) with a ROC curve area of 0.72. The combined mineralogy factor F_M, therefore, has the highest weighting. The fabric factor F_A has a ROC area of 0.69, and has the second highest weighting. The grain size and grain size distribution factor F_G has the lowest ROC curve area of 0.65, and therefore the lowest weighting.
4.4 Summary of Micromechanics and Rock Behaviour

4.4.1 Geological Characterisation for Spall Sensitivity

A literature review was conducted to determine the geological factors that were most important in controlling the failure behaviour of intact rocks. Three types of factors were identified: the mineralogy, grain size and grain size distribution, and the fabric type and intensity. These factors were combined into a Geomechanical Characterisation scheme that results in a quantification of the spalling sensitivity and fracture potential.

4.4.1.1 Mineralogy

The mineralogy was categorised according to major (i.e. quartz, olivine, feldspar, calcite, amphibole, and pyroxene) and accessory (i.e. biotite, muscovite, garnet, pyrite and magnetite) minerals. Each category uses the total and relative mineral contents, which are incorporated into the mineralogy factor $F_M$.

4.4.1.2 Grain Size and Grain Size Distribution

The grain size and grain size distribution were categorised according to three grain size ranges and three grain size distribution types (isotropic, seriate and bimodal). The categories are incorporated into the grain size and grain size distribution factor $F_G$.

4.4.1.3 Fabric Type and Intensity

The fabric type and intensity were categorised into four fabric types (mineral preferred orientation, schistosity, cleavage and gneissic banding) and three intensity categories based on microlithon spacing.

4.4.1.4 Geomechanical Characterisation Scheme

The F Factors, outlined in Table 4.4.1, were developed as part of the overall methodology to relate geological characteristics to rock mechanics demonstrated in Figure 4.4.1. This flow chart illustrates the geological characteristic associated with each F Factor and how these factors
are interpreted to provide geomechanical information for determination of spall sensitivity and fracture potential. The term spall sensitivity, $F_{SS}$, is used to describe the impact that mineralogy, grain size and intensity of foliation have on rock yield behaviour that could lead to sudden spalling at the excavation boundary, as described in Section 4.1. The spall sensitivity factor, combined with standard lab strength predictions, describes differences in rock yield behaviour under induced stress conditions during TBM excavation. For this purpose, $F_{FI}$ is used to describe the potential for fracture at the cutter scale and the entire tunnel face scale.

This methodology can be used to anticipate rock yield behaviour at excavation boundaries to make predictions for TBM performance. This system was created for crystalline rocks, not sedimentary rocks, due to the difference in characteristics of grain boundaries found in sedimentary rocks. Ultramafic, volcanic and highly altered ore rocks are also not considered due to limited data in these rock types. The following sections describe the development of the classification scheme from Figure 4.4.1 as a conceptual model.

Table 4.4.1. Description of geomechanical characterisation factors

<table>
<thead>
<tr>
<th>Factor</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>$F_M$</td>
<td>Mineralogy. Total and relative percentage of major minerals, $F_{MM}$, such as quartz, olivine, feldspar, calcite, amphibole, and pyroxene, and total and relative percentage of accessory minerals, $F_{MA}$, such as biotite, muscovite, garnet, pyrite and magnetite, are weighted for their low, medium or high impact on fracturing and spalling behaviour. The combination of the two results in a low, medium or high designation for the mineralogy factor.</td>
</tr>
<tr>
<td>$F_G$</td>
<td>Grain size and grain size distribution. Median grain size, $F_{GP}$, grain size reduction due to tectonic processes, such as subgrain formation and grain boundary migration, $F_{GT}$, and grain size distribution, primary or secondary resulting from tectonic deformation, $F_{GD}$, are designated low, medium or high, and are combined to result in a low, medium or high designation for grain size and grain size distribution impact on fracturing and spalling behaviour.</td>
</tr>
<tr>
<td>$F_A$</td>
<td>Anisotropy. Foliation type, $F_{AF}$, and foliation dimension, $F_{AD}$, in combination as $F_A$, are assigned a low, medium or high designation for impact on fracturing and spalling behaviour.</td>
</tr>
<tr>
<td>$F_{SS}$</td>
<td>Spall Sensitivity. $F_M$, $F_G$ and $F_A$ are combined to determine the low, medium or high sensitivity to isotropic spalling.</td>
</tr>
<tr>
<td>$F_{FI}$</td>
<td>Fracture Potential. Standard lab strength values and $F_{SS}$ are combined to determine the fracture potential of the rock, and normally manifests itself as a reduction of the lab strength value, representing excavation boundary strength: $F_{FI} = F_{SS} \times \text{lab strength}$</td>
</tr>
</tbody>
</table>
**4.4.2 Classification of the Geomechanical Characterisation Scheme for Chipping Performance**

The TBM performance data were used in Chapter 3 to categorize each rock sample characterized by the geomechanical characterisation scheme according to chipping performance and tunnel face stability. The Geomechanical Characterisation scheme was assessed using the chipping performance and tunnel face stability categories assigned to the rock samples.

**4.4.2.1 Summary of Chipping Performance Analysis**

The thresholds and weightings of each F Factor were modified and further defined from Section 4.2 according to chipping performance. The mineralogy factor, $F_M$, was found to have the highest impact on chipping performance, followed by the fabric factor, $F_A$, and the grain size and grain size distribution factor, $F_G$.

**4.4.2.2 Summary of Tunnel Face Stability Analysis**

The only samples that coincided with tunnel face instability contain fabric, and as such the mineralogy factor, $F_M$, and grain size and grain size distribution factor, $F_G$, could not be
evaluated for their impact on tunnel face stability. The analysis of the impact of fabric on tunnel face stability showed that rock with widely spaced fabric (lower intensity) is more likely to result in tunnel face instability than rock with closely spaced fabric (higher intensity) under the conditions in which the tunnel was constructed in the Southern Aar granite.
Chapter 5: Numerical Calibration of Geomechanical Characterisation Scheme

5.1 Introduction

As discussed in Chapters 2 and 3, extensile fracturing at excavation boundaries under compressive stress dominates the rock behaviour focused on in this research. The Geomechanical Characterisation scheme presented in Section 4.2 was developed from literature review and reflects the properties of rocks that have been identified for their impact on extensile fracturing. In order to apply the geomechanical characterisation scheme to TBM design and performance prediction it must be quantified by examining the relative impacts of each factor. Numerical modelling was selected as a complementary investigation to the parametric analysis of geomechanical characteristics on TBM performance in Section 4.3 because of its applicability to rock mechanics, the availability of numerical codes that can be used for parametric analysis and the control over input parameters available through numerical modelling compared with physical testing of rock. This chapter presents the approach to and results of parametric analysis of the F Factors, from the Geomechanical Characterisation scheme introduced in Chapter 4, performed using FLAC (Fast Lagrangian Analysis Code (Itasca Consulting Group Inc., 2007)).

5.1.1 Numerical Model Selection

The numerical modelling code FLAC was selected for calibration for a number of reasons, namely that it is a finite difference, time-stepping algorithm (see Appendix E.1) that also allows complex heterogeneity and strain-controlled loading paths to be modelled directly. This code provides increased control over all input parameters, including geometry, mesh size and shape, loading scheme and constitutive model, compared with another robust geotechnical finite element modelling code: Phase² (Rocscience Inc., 2005). The determination of initial and boundary conditions, such as material property heterogeneity, is very versatile using this numerical modelling code and is used to model not only the strength of the rock type, but also variability of strength and behaviour due to foliation and/or grain scale heterogeneity. Constitutive models can be specified to change during deformation; for example progressive strength loss with user-definable magnitudes of plastic strain accumulation. Many of the limitations of conventional finite difference models (i.e. they need regular geometries) are
overcome by the Itasca, Inc. software, making it comparable, in terms of applicability to a variety of problems, to the finite element codes widely used in civil engineering. The fracture mechanics code ELFEN (Rockfield, 2001), although highly applicable to rock and fracture mechanics, was not accessible during this research and it is less customisable than FLAC.

Two simple two-dimensional plastic models were created for parametric analysis: uniaxial compressive test (UCS) and Brazilian test (BTS) (see Appendix E.1 and E.2 for geometry and code). Both of these tests were selected for their simple geometry, the availability of closed-form solutions for development and validity testing and their representation of laboratory strength tests, to which modelled values could be compared. Both models are two-dimensional, leading to limitations and benefits. The limitations mostly concern the plane-strain formulation, which is not a true representation of laboratory tests, in particular for the uniaxial compression test. Benefits of two-dimensional modelling are simplicity of the code, computation speed and simplicity of the boundary conditions. The simplicity of the approach is critical to parametric analysis, where parameters can easily and quickly be changed and their impact can be clearly evaluated due to the low complexity of the interactions between stress, strain and strength. Diederichs (2003) showed that behaviour in plane strain can be representative of UCS behaviour up to the point of crack coalescence and interaction.

The F Factors evaluated with numerical modelling are those isolated in the Geomechanical Characterisation scheme (Section 4.2) to describe rock texture, namely: mineralogy, grain size and grain size distribution, and fabric. In order to investigate these parameters separately, the numerical model was designed to model individual minerals and their properties, within a composite polycrystalline rock. By manipulating the finite difference mesh geometry, and the input parameters describing the constitutive models of different minerals, each of the factors can be individually evaluated for impact on rock strength and spalling sensitivity relative to each of the other factors.

The selection of the finite difference element size for plastic models requires a balance of benefits and downfalls, and must take into consideration the finite difference algorithm. Each of the models presented in this chapter was tested for mesh size sensitivity, to demonstrate the mesh-dependency on element size and the difference in results when strain is localised in the smaller mesh-sized model (see Appendix E.2).

5.1.2 Calibration Approach

In order to model the parameters in the geomechanical characterisation scheme, a methodology was developed by which each parameter could be isolated and varied
independently. The methodology involves creating a representative constitutive model for each of three common rock-forming minerals: mica, quartz and feldspar, verified by a parametric analysis of the input values estimated from literature review. The mineral types are assigned to elements in the model, which are associated with each other through an algorithm created to simulate real crystal geometries and orientations based on real mineral properties. The selection of the crystal, or grain, geometries and orientations is based on textures of real rock types. The rock types used for numerical rock simulation are part of the dataset collected during field work. The textures are verified by photomicrograph comparison, summation of modal mineral percentage and queries regarding grain size and orientation. The modelling is an investigation of the contrasts between different textures, and is achieved by representative constitutive models using the most realistic values available in literature and sensitivity testing of the input parameters.

The models consist of UCS and Brazilian tests created in FLAC 2D. The models were created to represent realistic tests at 1:1 scale so that analytical solutions could be compared to the model solutions. The models were tested to ensure that geometry and boundary conditions, such as mesh size and loading velocity, respectively, were appropriate to the rock being modeled and minimised geometry and boundary condition dependencies. A full discussion of the modelling undertaken to test for these effects, and the resultant geometry and loading conditions selected for testing is found in Appendix E.2.

The model outputs consist of applied stress, comparable to Brazilian index strength and UCS strength, as well as location of fracture initiation with respect to mineral types, macro fracture development and shear strain increments along fractures, stress-strain curves, axial-lateral strain curves and modelled acoustic emissions counts. These data are used to investigate the effects of geological parameters on systematic damage initiation stress (as discussed in Diederichs et al. (2004) and Eberhardt et al. (1999), for example), interpreted as the stress level at which failure begins to occur in the modelled sample, and laboratory peak stress, interpreted as the stress level at which the sample has failed (where a single fracture has fully penetrated the sample, or multiple fractures have coalesced and destroyed the sample). The stress level at which each phenomenon occurs will vary with geological properties. Their ratios, as well as absolute values, are used to weigh the impact of the modelled geological factors on relative strength and fracture behaviour.

As discussed in Diederichs et al. (2004), the laboratory peak strength can be higher than the in-situ strength of the rock, and the lower boundary for in-situ rock strength is related to the systematic damage initiation threshold, which is independent of laboratory or in-situ conditions.
In-situ strength and behaviour depends on the geological characteristics, damage history and induced stress resulting from the excavation (Diederichs et al., 2004). In this research the stress at which damage initiates is analogous to the systematic damage initiation threshold of Diederichs et al. (2004) and is compared to the laboratory peak stress output by the model to investigate the impact of the F-Factors on fracture potential and the resulting spall sensitivity of the modelled rocks. Investigations of in-situ strength dependence on F-Factors and spall sensitivity are addressed in Chapter 6.
5.2 Determination of a Constitutive Model

5.2.1 Introduction

Two simple models representing two-dimensional Brazilian and UCS test setups are used first to develop the methodology for inputting the geological parameters obtained by the geological characterisation scheme, and second to investigate the relative influence of each parameter on rock strength and fracture initiation.

5.2.1.1 Constitutive Models

Constitutive models define the relationship between stress and strain in the elastic and non-elastic state. For an elasto-plastic model in FLAC with no weakening after yield, the input parameters include density, bulk and shear elastic stiffness, yield strength parameters, and dilation properties. Where weakening or strengthening is anticipated after yield (often called strain softening and strain hardening respectively), residual strength and strain dependent strength can also be input.

Four basic constitutive models are shown in Figure 5.2.1. Figure 5.2.1a shows the elastic model, in which case the strain undergone by a material in response to applied stress is linear and fully recoverable. Figure 5.2.1b shows an elastic perfectly plastic model in which a finite amount of recoverable strain is possible up to a peak stress at which point the material can no longer support additional stress and plastic (non-recoverable) strain is accumulated at constant stress. Figure 5.2.1c shows an elastic strain hardening material, which can also withstand elastic strain up to a peak stress, at which point the material can support additional stress, but the rate of strain per stress increment is larger. Figure 5.2.1d shows an elastic strain softening material, again with an elastic strain component and a peak stress at which point the material can only support lower stress with accumulated strain.
5.2.1.2 Constitutive Models for Minerals

The goal of the modelling phase described in this chapter is to quantify the F Factors for geomechanical characterisation. The approach consists of using simple models and conducting a parametric analysis of the various factors. The factors under consideration are mineralogy, grain size and grain size distribution and foliation. The latter two factors depend on the availability of valid mineralogical constitutive models. FLAC was designed primarily to accommodate whole-rock approximations, with some built-in variability in the input parameters for introduction of heterogeneity. This methodology has been successfully applied in numerical modelling (not only in FLAC) for investigations of rock failure in simple (Diederichs, 2003; DeBorst, 2002; Fang and Harrison, 2002; Tang and Kaiser, 1998; Zhu and Tang, 2004) and complex tests (Liu et al, 2002a; Liu et al., 2002b).

The goal of the numerical modelling is to investigate tensile fracture processes by focussing on initiators and receivers of tensile fractures (Figure 5.2.2). Whether a mineral acts as an initiator or receiver of tensile fractures will depend on its strength, stiffness and dilation parameters, its geometry and alignment with the induced stress field, and the physical and geometrical properties of the minerals adjacent to it. To undertake this type of investigation, mineral-specific input parameters instead of whole-rock analogues are required, similar to other published work (Li et al, 2003). For this reason, considerable constitutive model refinement was necessary in order to assign appropriate constitutive models and input parameters for minerals. Published mineral-specific strength data, as well as parametric analysis and comparison to real fracture behaviour, were used to determine valid constitutive models for use in the parametric analysis of the F-Factors for geomechanical characterisation.
5.2.2 Available Constitutive Models

Numerous built-in constitutive models are available in FLAC, but the four most applicable models are reviewed here. These are: Elastic model, Hoek-Brown model, Mohr-Coulomb model and Strain-Softening model. As shown in Figure 5.2.1a, the elastic model results in linearly increasing stress with recoverable strain. No peak strength is prescribed and the elements in the model are never allowed to fail. This is very useful for examining induced stress fields, but does not provide any information related to fracture processes and stress and strain changes resulting from yielding elements. In contrast, the latter three models are termed plastic models, in which non-recoverable strain is accumulated after the strength of the element has been overcome and the element has yielded. The difference between the three models lies in the input parameters and the post-peak behaviour of the model (see Appendix E.1).

The Mohr-Coulomb model requires cohesion, friction and tensile cut-off as strength parameters, whereas the Hoek-Brown model requires a UCS, and m and s parameters, which are related to friction and cohesion while tensile strength is not specified as an independent variable. Since this investigation focuses on tensile failure, and the Mohr-Coulomb model allows the user to specify a tensile cut-off independent of cohesion and friction, its formulation was selected for modelling. A built-in variability function makes it possible to input a mean and standard
deviation of constitutive model parameters from which the program will select values randomly while assigning the properties to domains. This variability is intergranular, meaning that the properties are the same within a mineral grain. A second level of variability was added to vary the strength parameters of elements within a grain from 80% to 120% of the mineral grain value to account for imperfect crystal formation. The post peak strength behaviour of the Mohr-Coulomb constitutive model is perfectly plastic for cohesion and friction (Figure 5.2.1b) and immediately drops to zero for tensile strength (Itasca Consulting Group Inc., 2001).

The rock types under investigation fail by dropping their strength components after yielding (Figure 5.2.1d), and require control of the post-peak strength parameters. In order to do this, the Strain-Softening model, which uses Mohr-Coulomb strength parameters, is used. In this model it is possible to prescribe strength parameters as a function of plastic strain in individual yielded elements (Itasca Consulting Group Inc., 2001). This allows the localisation of strain necessary to generate discrete fractures, both in shear and in tension. The location, orientation, initiation, propagation and attenuation of the fractures in the model provide the necessary insight into determination of initiators and receivers of fractures, as well as resulting in the anticipated macro fractures for the material under investigation.

The bilinear strain softening ubiquitous joint constitutive model in FLAC is used to model the anisotropic strength in mica grains. Although the bilinear functionality is not used, this model allows strain softening of the matrix and the ubiquitous joints, which is not possible with the normal ubiquitous joint constitutive model. This constitutive model assigns a matrix and a ubiquitous joint constitutive model to each element, where the matrix and ubiquitous joints have different strength parameters. The matrix is isotropic but the ubiquitous joints can be oriented. When the FLAC cycle reaches a mica element, it will first solve for the matrix stiffness and strength, adjust any relevant plastic corrections resulting from matrix yielding, and then solve for the ubiquitous joint strength for the new stress components. If yielding occurs on the ubiquitous joints the plastic strength corrections and resulting stress components, will be calculated (Itasca Consulting Group Inc., 2001). This will allow mica grains to be evaluated for failure perpendicular (matrix) or parallel to (ubiquitous joints) the weaker cleavage planes.

To accomplish strain softening, behaviour charts relating the decrease (or increase) of a strength property as a function of accumulated plastic strain are input as strength parameters into the constitutive model. The choice of the slope of the strength/plastic strain graph (Figure 5.2.3) is related to the material stiffness and Poisson’s ratio and, in the case of FLAC, through the shear modulus and bulk modulus (Ryder and Ozbay, 1991).
A complication arising from strength loss as a function of accumulated plastic strain is the localisation of strain. This results from the loss of strength after element yield and the increased likelihood of further plastic strain in the yielded element due to the lower strength criterion. As stated earlier, this phenomenon is desired for simulation of localised macro fractures within the modelled material, although this phenomenon may become extreme as element sizes are decreased. The plastic strain resulting from element node displacements (due to stress) is greater for smaller elements than for larger elements. In the case of strain-softening, extreme strain localisation is possible due to the compounding of strain localisation from post-yield strength loss and small element size. For this reason, the models were run at 1mm, 0.5mm and 0.25mm element size to monitor the sensitivity of the model to localisation due to element size (see Appendix E.2). A less than 0.5% difference was found in UCS and corresponding strain values between each mesh size configuration. The element size of 0.5mm was selected as a balance between small element size for mineral representation and practicable model run times.

### 5.2.3 Input Values for Strain Softening Constitutive Model

A set of peak physical, elastic and laboratory strength properties for quartz, feldspar and mica is shown in Tables 5.2.1 to 5.2.3. These published values were used as input values for peak portions of the strain-softening constitutive model in the UCS and Brazilian models in FLAC. Although the strain-softening constitutive model allows change in parameter values with plastic strain, the stiffness moduli are independent of plastic strain. The majority of the published values, such as elastic, shear and bulk modulus, density and Poisson’s ratio, can be used as-is in the constitutive model, but the cohesion, friction and tensile strengths are more difficult to
directly relate between laboratory test values and modelling values. Difficulties arise from the differences in scale at which laboratory testing is undertaken compared with the scale of individual mineral grains in rock, which are reproduced in the numerical model. The cohesion can be derived either with the UCS and the relationship between Hoek-Brown parameters and Mohr-Coulomb parameters (Hoek, 1990) or from the relationship between mode I (KIC) and mode II (KIIc) fracture toughness (Laqueche, Rousseau and Valentin, 1986), if one of them is known, using relationships similar to the one shown in Figure 5.2.4.

Table 5.2.1: Average physical, elastic and laboratory strength properties of quartz from various sources, used as the basis for calculating input strength and stiffness parameters for FLAC modelling. Sources with an asterisk were summarised in Lama and Vutukuri (1978).

<table>
<thead>
<tr>
<th>Mineral</th>
<th>Density g/cm³</th>
<th>Young’s modulus GPa</th>
<th>Pidgidity (shear) modulus GPa</th>
<th>Bulk modulus GPa</th>
<th>Poisson’s ratio</th>
<th>Ucs MPa</th>
<th>Tensile strength MPa</th>
<th>Kic MPa</th>
<th>Internal friction</th>
<th>Source</th>
<th>Notes</th>
</tr>
</thead>
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<td>Quartz</td>
<td>2.72</td>
<td>1.71</td>
<td>2.8</td>
<td>58</td>
<td>22.8</td>
<td>0.2</td>
<td>328.86</td>
<td>33.78</td>
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<td>2.59</td>
<td>53.8</td>
<td>2.47</td>
<td>37.9</td>
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<td>36</td>
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<td>0.3625</td>
<td>1.6</td>
<td>Birch, 1966</td>
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<td>0.25</td>
<td>Broz et al., 2006</td>
<td>Estimates from Vickers indentation test for quartz, fused quartz (fiberoptic glass)</td>
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<td></td>
<td></td>
<td></td>
<td>253.54</td>
<td></td>
<td>Lama &amp; Vutukuri, 1978, various sources</td>
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<td>0.90</td>
<td>Tromans &amp; Meech, 2004</td>
<td>Alpha quartz, theoretical values</td>
<td></td>
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</tbody>
</table>

* average does not include value from Blair 1955 (assumed erroneous) or porphyry from Cochrane 1964 (assume mineral damage leads to low values)
Table 5.2.2: Average physical, elastic and laboratory strength properties of feldspar from various sources, used as the basis for calculating input strength and stiffness parameters for FLAC modelling. Sources with an asterix were summarised in Lama and Vutukuri (1978).

<table>
<thead>
<tr>
<th>Mineral</th>
<th>Density g/cm³</th>
<th>Young’s modulus (shear) GPa</th>
<th>Rigidity modulus Gpa</th>
<th>Bulk modulus GPa</th>
<th>Poisson’s ratio</th>
<th>Ucs MPa</th>
<th>Tensile strength MPa</th>
<th>Kic</th>
<th>Internal</th>
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<tr>
<td>Feldspar</td>
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<tr>
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<td>2.61</td>
<td>76.1</td>
<td>30</td>
<td>0.27</td>
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<td></td>
<td>0.88</td>
<td>Broz et al, 2006</td>
<td>estimates from Vickers indentation test</td>
<td></td>
</tr>
<tr>
<td></td>
<td>2.57</td>
<td>71</td>
<td>34.6</td>
<td>0.28</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>*Belkio, 1967</td>
<td>albite</td>
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</tr>
<tr>
<td></td>
<td>2.7</td>
<td>88.7</td>
<td>34.6</td>
<td>0.28</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>*Belkio, 1967</td>
<td>labradorite</td>
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<td></td>
<td>2.7</td>
<td>89</td>
<td></td>
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<td></td>
<td></td>
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<td>*Belkio, 1967</td>
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<tr>
<td></td>
<td>2.55</td>
<td>74.9</td>
<td>26.3</td>
<td>0.27</td>
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<td>*Belkio, 1967</td>
<td>microcline</td>
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<td></td>
<td>2.01</td>
<td>77.3</td>
<td>29.8</td>
<td>0.25</td>
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<td></td>
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<td>*Belkio, 1967</td>
<td>oligoclase, 22% anorthite</td>
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<td></td>
<td>2.54</td>
<td>63</td>
<td>24.4</td>
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<td>*Belkio, 1967</td>
<td>oligoclase, mineral</td>
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<td>95.6</td>
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<td>Angel et al, 1988</td>
<td>orthoclase end-member feldspars</td>
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<td>Birch, 1966</td>
<td>orthoclase, initial compressibility</td>
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<td>52</td>
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<td>40.5</td>
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<td></td>
<td>Birch, 1966</td>
<td>initial compressibility</td>
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<td>57</td>
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<td></td>
<td></td>
<td>Troianos &amp; Megach, 2002</td>
<td>anorthite, theoretical values</td>
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<td></td>
<td></td>
<td></td>
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<td>Trench, 2002</td>
<td>anomomosts</td>
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<td>2.61</td>
<td>62.25</td>
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<td>51.38</td>
<td>0.26</td>
<td>200.06</td>
<td>0.06</td>
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<td>Lama &amp; Vutukuri, 1978, various sources</td>
<td>syenite</td>
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<tr>
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<td>average</td>
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<td></td>
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</tr>
</tbody>
</table>

* not enough data for directionality with respect to cleavage planes
Table 5.2.3: Average physical, elastic and laboratory strength properties of mica from various sources, used as the basis for calculating input strength and stiffness parameters for FLAC modelling. Sources with an asterix were summarised in Lama and Vutukuri (1978).

<table>
<thead>
<tr>
<th>Mineral</th>
<th>Density g/cm³</th>
<th>Young's modulus GPa</th>
<th>Rigidity modulus GPa</th>
<th>Bulk modulus GPa</th>
<th>Poisson's ratio</th>
<th>Ucs MPa</th>
<th>Tensile strength MPa</th>
<th>Kic MPa</th>
<th>Internal friction Notes</th>
</tr>
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<tbody>
<tr>
<td>Mica</td>
<td>2.75</td>
<td>20</td>
<td>150</td>
<td>0.28</td>
<td>178.6</td>
<td>76.8</td>
<td>52.6</td>
<td>0.26</td>
<td>*Comins &amp; Nestott 1967 biotite in gneiss</td>
</tr>
<tr>
<td></td>
<td>3.1</td>
<td>69.7</td>
<td>27.7</td>
<td>0.28</td>
<td>52.7</td>
<td>5.6</td>
<td>52.6</td>
<td>0.26</td>
<td>*Belkoff, 1967 biotite</td>
</tr>
<tr>
<td></td>
<td>178.6</td>
<td>76.8</td>
<td>52.6</td>
<td>0.26</td>
<td>147</td>
<td>68</td>
<td>147</td>
<td>0.15</td>
<td>Birch, 1966 muscovite, parallel to basal plane</td>
</tr>
<tr>
<td></td>
<td>2.78</td>
<td>60.9</td>
<td>112</td>
<td>0.15</td>
<td>176.5</td>
<td>70</td>
<td>60.9</td>
<td>0.15</td>
<td>*Coates &amp; Parsons 1966 chlorite with 10-30% calcite</td>
</tr>
<tr>
<td></td>
<td>176.5</td>
<td>70</td>
<td>60.9</td>
<td>0.15</td>
<td>60.9</td>
<td>15</td>
<td>60.9</td>
<td>0.15</td>
<td>Moneil &amp; Grimsditch, 1993 muscovite, parallel to basal plane</td>
</tr>
<tr>
<td></td>
<td>60.9</td>
<td>15</td>
<td>60.9</td>
<td>0.15</td>
<td>0.15</td>
<td></td>
<td></td>
<td></td>
<td>Atkinson &amp; Meredith, 1987 muscovite, calculated from fracture surface energy, splitting parallel to basal plane</td>
</tr>
<tr>
<td></td>
<td>average</td>
<td>2.66</td>
<td>97.74</td>
<td>38.46</td>
<td>51.99</td>
<td>0.26</td>
<td>131.00</td>
<td>0.15</td>
<td>167.37</td>
</tr>
<tr>
<td></td>
<td>average parallel</td>
<td>55.53</td>
<td>10.93</td>
<td></td>
<td>2.11</td>
<td></td>
<td></td>
<td></td>
<td>71.60</td>
</tr>
</tbody>
</table>

Figure 5.2.4: Relationship between $K_{IC}$ and $K_{IIC}$ and normalised notch length where $a/w$ is the ratio of initial fracture length to sample thickness, Laqueche et al. (1986).
Fracture toughness is a property that expresses the resistance to catastrophic crack propagation (Whittaker et al., 1992). The $K_{IC}$ for rocks will vary greatly due to the heterogeneous nature of polycrystalline and polymineralic substances. In contrast, the $K_{IC}$ for an individual mineral should be intrinsic to that mineral since the fracture toughness of a single mineral should be directly related to its chemical composition and crystal lattice. $K_{IC}$ testing on quartz and feldspar was conducted by Vickers Indentation and theoretical calculation (see Tables 5.2.1 - 5.2.2), while for mica the $K_{IC}$ is calculated from the basal plane fracture surface energy and the basal plane Young’s Modulus for muscovite (Table 5.2.3). These values are used to interpret the relationship between input tensile strength values and test model output fracture toughness values. For this purpose an investigation of the relationship between model input tensile strength and output fracture toughness is used to determine the necessary input tensile strength, and by extension, better estimate cohesion values.

The peak strength and failure coefficient of whole rocks is dependent on the friction coefficient of the constituent minerals. While the friction coefficient for intact minerals can be very low (Byerlee, 1978), the post-peak friction coefficient increases considerably once plastic strain has been accumulated. The friction coefficients of mono-mineralic gouges were used as proxies to estimate large-strain post-peak friction coefficients.

Quartz is assumed to be isotropic in its stiffness and strength, and while feldspar has some anisotropy, it is treated as isotropic in stiffness and strength due to lack of directionality data. There is sufficient directionality data for mica (as discussed in Chapter 4.2.2) to be treated as a transversely anisotropic mineral in stiffness and in some strength parameters, although for simplified preliminary numerical model runs an isotropic average value can be used. The terms parallel and perpendicular are used according to the direction in which the maximum principal stress will act on an individual mineral grain, as shown in Figure 5.2.5. Note that strength is weakest parallel to the basal planes, while stiffness is lowest perpendicular to the basal planes. Poisson’s ratio also varies depending on the direction of loading, but the bulk modulus is a function of all three dimensions and is the same in any loading direction (it is calculated from a hydrostatic stress condition) and shear modulus is the same in both directions since both are a result of the combination of the stresses parallel and perpendicular to the basal planes. A third direction exists in micas: the direction parallel within the basal planes, which has much higher shear modulus (as shown in Table 5.2.3), Young’s modulus identical to the parallel case and similar Poisson’s ratio. Since the minerals are modelled in two dimensions, and the two most likely directions of deformation in three dimensions are represented by Figure 5.2.5, then this third direction is not considered for modelling.
5.2.3.1 Stiffness Moduli

The stiffness moduli for minerals are listed in Tables 5.2.1 - 5.2.3, and are assumed to be independent of plastic strain. To support the validity of these values, the bulk moduli for quartz, orthoclase and labradorite were calculated using Young’s modulus and Poisson’s ratio, according to:

\[ K = \frac{E}{3(1-2\nu)} \]  \hspace{1cm} (5.2.1)

The ratios between bulk moduli of two minerals calculated from the average values in Tables 5.2.1 - 5.2.2 and volumetric compression (from Clark, 1966 (Birch, 1966) in Lama and Vutukuri (1978)) values were compared by using the relationship that for a constant stress:

\[ \frac{\Delta V_i}{V_i} \propto \frac{1}{K_i} \] \hspace{1cm} (5.2.2)
and:
\[
\frac{\Delta V}{V^2} \propto \frac{1}{K_i}
\]

Rearranging gives:
\[
\frac{\Delta V}{V^i} = K_i
\]

Each side of Equation 5.2.4 was calculated separately for quartz and orthoclase, and quartz and Labradorite, and compared. It was found that equation 5.2.4 holds true for both sets of tests, independently demonstrating that the Young’s modulus and Poisson’s ratio values for quartz and feldspar (orthoclase and labradorite, in this example) from Belikov (1967) in Lama and Vutkuri (1978) are reasonable (Table 5.2.4).

The values in Tables 5.2.1 - 5.2.2 show some variability, which is taken into consideration by the range of values listed in Table 5.2.11. These are used in the numerical models as second standard deviation values for normal distributions of the stiffness moduli.

The stiffness moduli for biotite were calculated using elastic constants for the crystal lattices (Section 4.2.2.4.2), and demonstrate a large anisotropy due to the anisotropy along and against the basal planes. The validity of these values was not corroborated with independent values as no such values are available, but the fundamental nature of the equations attests to their validity and they were used as-is. Due to the extreme anisotropy, the stiffness moduli relationships (i.e. equation 5.2.1) do not hold true for micas, and the shear modulus and bulk modulus must be calculated using the elastic constants (Section 4.2.2.4.2). The anisotropy in stiffness is only addressed by the Young’s modulus, which is not a parameter in FLAC, while both shear and bulk modulus are the same in either direction in 2-D.

Table 5.2.4: Comparison table showing results of testing ratios between volumetric strains and ratios between bulk moduli for combinations of quartz and two feldspars: orthoclase and Labradorite. The ratios are within less than 1% of each other, substantiating the validity of the values derived from two sources.

<table>
<thead>
<tr>
<th></th>
<th>Quartz</th>
<th>Orthoclase</th>
<th>Labradorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Young’s Modulus (GPa)</td>
<td>64.4</td>
<td>63</td>
<td>88.7</td>
</tr>
<tr>
<td>Poisson’s Ratio</td>
<td>0.19</td>
<td>0.29</td>
<td>0.28</td>
</tr>
<tr>
<td>Bulk Modulus (GPa)</td>
<td>34.5</td>
<td>50</td>
<td>67.2</td>
</tr>
<tr>
<td>Volumetric strain</td>
<td>0.0236</td>
<td>0.0171</td>
<td>0.0133</td>
</tr>
<tr>
<td>Ratios versus quartz</td>
<td>Volumetric strains</td>
<td>1.38</td>
<td>1.77</td>
</tr>
<tr>
<td></td>
<td>Bulk Moduli</td>
<td>1.45</td>
<td>1.95</td>
</tr>
</tbody>
</table>
5.2.3.2 Fracture Toughness and Relationship to Tensile Strength

In order to better understand how FLAC uses tensile strength input values for constitutive models in terms of tensile fracturing, a fracture toughness numerical model was created (Appendix E.2). The numerical model was used to compare input tensile strength parameters with the output fracture toughness, to identify what the relationship is in FLAC and how this relationship compares to published values.

Published relationships between laboratory $K_{IC}$ and tensile strength values from rocks suggest that the tensile strength can be estimated from the $K_{IC}$ as follows:

$$\sigma_t = aK_{IC}$$  \hspace{1cm} 5.2.5

where $a$ is a constant (units $[L]^{-1/2}$) derived from regression of test data, and has been found to be 8.66 (Zhang, 2002) and 9.35 (Whittaker et al., 1992), for rocks, for example. The results of fracture toughness numerical modelling below show that the $a$ constant is approximately 37-41 for quartz, 40-43 for feldspar and 35-40 for mica.

5.2.3.2.1 Model Set-up

The outcomes from sensitivity analysis conducted during model development were used to determine the geometry and loading rate used in this investigation, as described in Appendix E.2. The data from Figure 5.2.4 were recreated with results from numerical modelling of the three point bending beam test with increasing initial notch lengths (Figure 5.2.6 a and b) in which the arrows show reaction forces (opposing applied velocity) at the sample top and reaction forces at the fixed points at the sample base. The results for $K_{IC}$ and $K_F$, the critical stress intensity factor and stress intensity factor at first element yield respectively, are plotted versus initial notch length in Figures 5.2.6c and 5.2.7. Both of these plots have similar trends as the plot in Figure 5.2.4, suggesting that the numerical model is a valid representation of the physical test. Not to be confused with the testing conducted by Laqueche et. al. (1986) and in Figures 5.2.6 and 5.2.7, it has been found that $K_I$ decreases during fracture extension from the initial notch, for any initial notch length, which was also seen by Lévay et. al. (2003) for polyamide. This relationship arises from the lower force required to propagate the fracture once it has been initiated, compared to the force required to initiate the fracture. This is due to the energy accumulated in the system prior to initiation, which is then released once the first elements yield and is then transferred to the unyielded elements, a process similar in principle to the fracture process zone discussed in Whitaker et al (1992).
Figure 5.2.6: Schematic of three-point bending beam test with close-up of changes in notch length (a); FLAC output of grid with reaction force (opposing applied velocity at sample top, reaction forces at the fixed points at the sample base) and close up of notch (b); Comparison of $K_\text{IC}$ and corresponding displacement with different notch lengths, loaded at two different model velocities: $7\times10^{12}$ and $9\times10^{12}$ (c).
The $K_{IC}$ value is very sensitive to the initial notch length (Okubo and Fukui, 1996; Whittaker et al., 1992) due to the formulation of the $K_{IC}$ equation, making the values for the initial notch length=0 invalid because $\alpha = 0$, giving a $K_{IC} = 0$. The peak load plot in Figure 5.2.7 demonstrates the decreased load requirements in samples with a stress concentrator (initial notch length). In the plots in Figures 5.2.6 and 5.2.7, a plateau exists between normalised notch length of $\alpha = 0.1-0.4$, similar to the graph in Figure 5.2.4. For this reason, an initial notch length $\alpha = 0.3$ (Figure 5.2.6 b) was used to determine the model input tensile strength. This model represents a mineral that has a pre-existing stress concentrator (i.e. a flaw) equal to one third of the mineral dimension.

Figure 5.2.7: Comparison of $K_f$ and peak load with different initial notch lengths, loaded at two different model velocities: $7e12$ and $9e12$
5.2.3.2 Tensile Strength Testing

Appropriate mineral-specific tensile strengths were determined using the three-point bending beam test (configured as shown in Figure 5.2.6 b) as an analogue to individual mineral grains. A homogeneous tensile strength was input into a strain-softening constitutive model for the test. The resulting $K_{IC}$ and $K_F$ values were recorded and compared to published values in Tables 5.2.1 to 5.2.3. An iterative process was used to obtain the published $K_{IC}$ value for quartz and the required input tensile strength was recorded. This is the strength FLAC requires to simulate the critical stress intensity factor of a homogeneous, mono-mineralic beam. The relationship for quartz was found to be linear. Figures 5.2.8 to 5.2.10 show the model $K_{IC}$ and $K_F$ values corresponding to FLAC input tensile strength values for quartz, feldspar and mica, and the selection of the appropriate FLAC input tensile strength for rock modelling, shown in Table 5.2.11. The FLAC input tensile strength may or may not be the true tensile strength of the mineral, since this has not been verified by physical mineral-specific testing, but it is the value required for FLAC to model the yielding behaviour of the minerals in question.

The $K_{IC}$ output values are dependent on input stiffness values (bulk and shear modulus) (Chang et al., 2002; Whittaker et al., 1992), which was also substantiated by testing with very high stiffness material. For this reason, the appropriate bulk and shear stiffness moduli for each mineral were used in testing. The cohesion and friction parameters were arbitrary as the tensile yielding in the three-point bending beam model was found to be independent of cohesion and friction.

These results show that the ratio between input tensile strength and output $K_{IC}$ is approximately 37-41 for quartz, 40-43 for feldspar and 35-40 for mica. This results in input tensile strength values of 15MPa, 16MPa and 4.5MPa for quartz, feldspar and mica, respectively, based on the Vickers indentation tests (Atkinson and Avdis, 1980; Broz, Cook and Whitney, 2006) and $K_{IC}$ values listed in Tables 5.2.1 - 5.2.3. Those from theoretical values based on mineral lattice strength (Tromans and Meech, 2002; Tromans and Meech, 2004) in Tables 5.2.1 - 5.2.3 result in input tensile values of 29MPa and 33MPa for quartz and feldspar, respectively. The wide range in input values suggests that these values are not necessarily more reliable than the tensile strength values also listed in Tables 5.2.1 for quartz. This set of values provide a lower and upper bound for the second standard deviation of the FLAC input tensile strength normal distribution.
Figure 5.2.8: Linear relationship between input tensile strength and output fracture toughness values for quartz used to determine the appropriate input tensile strength for quartz.

Figure 5.2.9: Input tensile strength and output fracture toughness values for feldspar used to determine the appropriate input tensile strength for feldspar.
Figure 5.2.10: Input tensile strength and output fracture toughness values for mica used to determine the appropriate input tensile strength for mica.

The stiffness of micas was shown in Section 5.2.3.1 to be highly anisotropic, in particular in the directions parallel and perpendicular to the mineral basal planes. The same should hold true for fracture toughness and tensile strength. Although no published data for micas can substantiate this, graphite, another platy mineral with extreme stiffness and strength anisotropy due to its perfect cleavage, has a 1:10 ratio of parallel to perpendicular fracture toughness (Tromans and Meech, 2004). If graphite can be used as an analogy for mica, then a similar ratio between parallel and perpendicular fracture toughness should also exist for mica. The strong covalent bonding between mica tetrahedra and octahedra (McNeil and Grimsditch, 1993) is similar to the very strong covalent bonding between carbon atoms in graphite, suggesting that the ratio between parallel and perpendicular fracture toughness may approach 1:10, resulting in a maximum input perpendicular tensile strength of 39MPa.

### 5.2.3.3 Friction Determination

Two types of friction are commonly reported in the shear failure of rock: initial friction and maximum friction (Byerlee, 1978), where the initial friction is shown to be lower than the maximum, and residual, frictional strength of rock (Figure 5.2.11). The initial friction is related
to the onset of plastic deformation, and represents the initiation of damage. The maximum friction is the static friction that must be overcome for two failure planes to begin to move past each other, while the residual friction remains after continued displacement has occurred and is considered the kinetic friction. The coefficient of friction is a property of the material, regardless of the surface roughness, and relates frictional force to normal force (Byerlee, 1978; Horn and Deere, 1962).

The initial coefficient of friction (A in Figure 5.2.11) is likely an artefact of the testing methodology, especially in the case of mineral gouges, where some displacement is measured due to reseating of mineral fragments. This would result in early displacement and an apparent friction increase with strain due to locking up of fragments and mobilisation of true friction. For this reason, only two values for friction coefficient from Table 5.2.5 are used: the mineral surface values and the static gouge values.

Figure 5.2.11: Frictional force (or coefficient of friction, if presented independent of normal force) versus displacement schematic from shear testing showing the locations of: A – initial friction, the point where plastic displacement occurs; B – maximum friction; C – residual friction; and D: stick-slip, where the sample jumps due to the release of built-up elastic energy. Several episodes of stick slip may occur, or the sample may deform by stable shearing, represented by the dotted line. (Byerlee, 1978).
The maximum, or static, friction appears to be independent of broad rock type, i.e. metamorphic, sedimentary and igneous, and initial friction appears to be independent of specific rock type or strength, i.e. granite versus tuff, and lies around $\mu=0.85$ (Byerlee, 1978).

Experiments on polished mineral surfaces (Horn and Deere, 1962) and monomineralic fault gouges has shown that friction (initial and static) is not independent of mineral type, i.e. quartz vs muscovite (Morrow et al., 2000), and that initial and static friction differ according to the schematic in Figure 5.2.11.

The internal friction coefficient measured by ultrasonic pulses (Fukuhara et al., 1997), in the case of quartz, is comparable to the dry static friction coefficient (Table 5.2.5) of single minerals. The value determined by Fukuhara et al. (1997) at 273° K compares well with the values determined by Horne and Deere (1962) for dry static friction of polished mineral surfaces, but both are much lower than the dry initial and dry static friction coefficients of mineral gouge. The same holds true for feldspar, but the difference in values for the platy minerals (comparing biotite to chlorite and muscovite) is less pronounced. In modelling mineral failure, the mineral lattice friction is mobilised once the stress inside the grains has reached its peak strength, and provides strength to a deforming mineral microfracture. With larger deformation and damage to the mineral surfaces, the mineral is more akin to a gouge, and its friction coefficient is likely similar to the values determined by Morrow et al. (2000). This is not the same as macro-fracture properties of composite rock, for which coefficients of friction are much higher than for minerals. For this reason, two friction coefficient values are used to reflect this change in phase: the mineral surface coefficient of friction measured by Horn and Deere (1962) to simulate the lattice friction, and the friction coefficient of gouge to simulate damaged mineral friction (Figure 5.2.12). In the case of biotite, the ratios from muscovite and chlorite are used to estimate the polished mineral surface to gouge friction coefficient ratio, and thus, the mineral gouge friction coefficient.

Table 5.2.5: Friction coefficient values for quartz, feldspar and platy minerals commonly found in study rocks, from three different authors.

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Quartz</td>
<td>0.13</td>
<td>0.12</td>
<td>0.55</td>
<td>0.65</td>
</tr>
<tr>
<td>Feldspar</td>
<td>0.12</td>
<td>0.55</td>
<td>0.8</td>
<td></td>
</tr>
<tr>
<td>Biotite</td>
<td>0.31</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Muscovite</td>
<td>0.43</td>
<td>0.2</td>
<td>0.6</td>
<td></td>
</tr>
<tr>
<td>Chlorite</td>
<td>0.53</td>
<td>0.2</td>
<td>0.68</td>
<td></td>
</tr>
</tbody>
</table>
Figure 5.2.12: Schematic of plastic strain dependence of friction coefficients of mineral surface phase and mineral gouge phase, showing lattice, $\phi_l$, and base, $\phi_b$, coefficients of friction.

The static coefficient of friction values for minerals are lower than the average value for rocks determined by Byerlee (1978), suggesting that although the friction coefficient is independent of roughness in theory, in practice, the measured frictional force is dependent on the coefficient of friction, the impact of macro scale interlocking asperities and effect of polycrystalline interaction in the formation of macrofractures. Nevertheless, roughened mineral surfaces were found to have higher coefficients of friction than their polished counterparts (Horn and Deere, 1962), suggesting that mineral-specific coefficients of friction may have an additional term to account for the interlocking of asperities due to microfracture scale roughness. In the case of rock, this term is normally called $i$ and is the angle of inclined surfaces over which the fractures must slide during displacement, measured as the ratio of the amplitude to the wavelength of the fracture surface (Patton, 1966).

Intragranular microfractures are assumed to have the same friction coefficient to roughness coefficient, $i$, relationship as macrofractures in polycrystalline rock. Equation 5.2.6 is used to combine both aspects of friction and is assumed to be valid at low strains, where failure through the asperities is not of concern. This assumption is not to be confused with relationships governing microfractures or macrofractures through polycrystalline rock.

$$\tau = \sigma_n \tan(\phi_b + i)$$  \hspace{1cm} 5.2.6

where $\sigma_n$ is the normal stress, $\phi_b$ is the basic friction angle and $i$ is the angle of the saw-tooth fracture surface (Patton, 1966). The mineral base friction is, unlike the basic friction from...
equation 5.2.6, made up of the lattice friction and the microfracture surface roughness coefficients (Figure 5.2.12):

\[ \phi_b = \phi_i + i_l \quad 5.2.7 \]

An analysis of the ratio between amplitude and length of the microfractures present in the quartz grains (Figure 5.2.13) of Stanstead granodiorite was conducted and shows that the \( i_l \) ranges from 1-1.5°, giving a \( \tan(i_l) \) 0.018 to 0.028. This is only slightly more than a 10% increase in the base friction coefficient value given in Table 5.2.6. Figures 5.2.14 and 5.2.15 show the nature of intragranular microfractures in mica and feldspar, in Leventina gneiss, which tend to be parallel to, or perpendicular to cleavage, which in all images in Figure 5.2.14 is parallel to the long axis of the mineral grain. The cleavage parallel microfractures are very smooth, giving an \( i_l \) of 0, while the cleavage perpendicular microfractures follow a more tortuous path (Figure 5.2.14 b and c), giving an \( i_l \) of approximately 25° and a \( \tan(i_l) \) of 0.47. Where cleavage parallel and perpendicular microfractures in mica coalesce, they form a microfracture with large amplitude (Figure 5.2.14 a), giving an \( i_l \) of approximately 15°, and a \( \tan(i_l) \) of 0.27. According to these measurements, the mineral base coefficient of friction is lower than the gouge coefficient of friction for quartz and cleavage parallel microfractures, or higher than the gouge coefficient of friction for cleavage perpendicular microfractures.

The obstruction to shear strain during mineral failure arising from tortuous intragranular microfractures suggests that the resultant ‘friction angle’ is much higher than \( i_l \). This value would be very difficult to measure, but evidence supporting this hypothesis was found by Diederichs (1999), where an input grain-specific friction angle of 45° was necessary to obtain a whole-rock friction angle approaching 30° during biaxial testing in 2-D particle flow code (PFC) models. Based on these findings, a nearly instantaneous (at \( \varepsilon = 1 \times 10^{-8} \)) ‘friction angle’ gain of 45° is used in 2-D FLAC models. This simulates the instantaneous transition from a continuous array of mineral elements to the discontinuity caused by fracture generation. The initial friction remains the same as that quoted in Table 5.2.6 to ensure that the element failure occurs at the appropriate stress level, as investigated using acoustic emissions from the FLAC model (1 element failure = 1 acoustic emission). The instantaneous increased friction angle simulates the initial interlocking friction at steep discontinuity asperity slopes. Although fractures generated along the cleavage mica plane have rare asperities and should not have instantaneous friction gain, in order to prevent numerical instability caused by intense strain localisation along ubiquitous joints, an instantaneous increased friction angle is also applied to the mica ubiquitous joints. The friction angle recovers to the gouge friction value according to the strain rate described earlier. The cohesion does not drop to residual at the same strain rate as the friction increase to maintain
numerical stability that is likely with the extreme strain localisations that would occur if cohesion were to drop quicker than the strains can be redistributed within the matrix. In addition, a strain-softening rate that is too fast would not allow dilation to be activated during element failure.

Figure 5.2.13: Photomicrographs of intragranular quartz microfractures in Stanstead granodiorite used to determine the microfracture roughness, \( i_i \).

Figure 5.2.14: Photomicrographs of mica (a, b) and feldspar (c) in Stanstead granodiorite showing microfractures used to determine microfracture roughness, \( i_i \).
Figure 5.2.15: Photomicrograph of feldspar microfractures parallel to cleavage in Stanstead granodiorite.

Table 5.2.6: Mineral-specific friction coefficients based on published (Horne and Deere, 1962; Morrow et al., 2000) and calculated values.

<table>
<thead>
<tr>
<th>Mineral</th>
<th>Individual Minerals</th>
<th>Mineral Gouges</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>$\mu_{b}$</td>
<td>$\mu_{G}$</td>
</tr>
<tr>
<td>Quartz</td>
<td>0.12</td>
<td>0.65</td>
</tr>
<tr>
<td>Feldspar</td>
<td>0.12 – 0.47</td>
<td>0.8</td>
</tr>
<tr>
<td>Biotite</td>
<td>0.31 – 0.78</td>
<td>0.43</td>
</tr>
</tbody>
</table>

Due to the anisotropy in the microfracture roughness coefficient, the cleavage (basal plane) parallel and perpendicular values for mica are associated with the corresponding strength and stiffness parameters according to Figure 5.2.5. While the same could be done for feldspar, insufficient directional data exists for the strength parameters to implement this, and the high and low roughness coefficient values are used as low and high end members of a normal distribution.

5.2.3.4 Dilation Parameter Determination

Although brittle rock failure is highly dilatant (Diederichs, 2003), the determination of the dilation parameter for brittle rocks is difficult and anisotropic. Several rules of thumb exist for isotropic dilation, including the associated flow rule (where the dilation parameter is equal to
the coefficient of friction) in both tensile and shear failure (Itasca Consulting Group Inc., 2001). They are all based on continuum models and are not directly applicable to brittle rock failure. Dilation can range from 0 to the coefficient of friction according to continuum model methods and determining the exact value is still unclear. For the purposes of this research, a dilation angle of 15% of the initial friction angle is for tensile, uniaxial and biaxial failure.

5.2.3.5 Cohesion Determination

Mode I fracture is related to tensile strength, as shown in Section 5.2.3, and mode II fracture is a type of shear failure and would, therefore, be related to cohesion. In contrast to mode I fracture toughness (K_I), mode II fracture toughness (K_II) is insensitive to initial notch length (Whittaker, Singh and Sun, 1992), which can be seen in Figure 5.2.4. The ratio between K_I and K_II will then also depend on the selected initial notch length. The initial notch length used in the K_I testing falls on the plateau with respect to initial notch length, Figures 5.2.6 and 5.2.7, making the ratio between K_I and K_II applicable to the majority of normalised initial notch lengths, except for the very small lengths (α < 0.1).

Since little work has been conducted on pure mode II failure, relationships between K_II and K_I can only be revealed for mixed-mode I-II failure (Whittaker et al., 1992). According to the Griffith failure criterion, the K_II/K_I ratio should be between 0.627 and 0.866 for mixed-mode I-II failure, although the limited mode II data that is available, show K_II to be higher than K_I (Whittaker et al., 1992).

The K_II values of polycrystalline ceramics are typically higher than for K_I (Laqueche et al., 1986; Whittaker et al., 1992) due to resistance at the fracture surface from grain interlocking and abrasion (Singh and Shetty, 1989). Polycrystalline ceramics have been shown to exhibit higher K_II than monocrystalline soda lime glass (Singh and Shetty, 1989). By applying this logic to rock, the K_II of polycrystalline rock should be greater than the K_II of individual minerals. A compilation of K_II/K_I values for different materials (rock and ceramics) in Whitakker et al. (1992) gives a range of 0.61-2.02. Typical granite values fall between 1.01-1.13 while values for syenite range between 0.61-0.675. The method by which cohesion or friction angle could be derived from K_II values is not clear, however, and would require a series of numerical models of a K_II test to identify the relationship between input cohesion and output K_II. Since no standard test exists for this (Whittaker et al., 1992) the analysis would not be reliable.
The second method of determining cohesion is using the relationship between UCS, cohesion and coefficient of friction, according to:

\[ c = \frac{UCS(1 - \sin \phi)}{2 \cos \phi} \]  \hspace{1cm} 5.2.8

Using the published values for UCS for minerals, as well as mono-mineralic rocks, in Tables 5.2.1 to 5.2.3 and the base friction coefficient values for each mineral, calculated from the values in Table 5.2.6 using Equation 5.2.8, an estimate of peak cohesion is obtained. Since cohesion and the UCS are related to each other through the friction coefficient, one can be calculated from the other two as their values change throughout the yielding process during fracture generation. In a purely unconfined state the frictional component of strength would not be mobilized and cohesion would be estimated as half of the UCS strength, suggesting that by using Equation 5.2.8 to determine cohesion, it is likely underestimated. The cohesion, however is a rock property as defined by the Mohr-Coulomb methodology, and independent of confinement. The use of Equation 5.2.8 is, therefore, the most appropriate approach to estimating the basic mineral strength parameters.

If the cohesion is calculated as a function of decreasing mineral UCS strength and increasing friction, three basic values for cohesion are used in the strain-softening constitutive model, reflecting the changing coefficient of friction as the mineral strains from base friction to the gouge “residual” friction, once the mineral has changed from single mineral phase to gouge phase due to deformation (as in Figure 5.2.11). Cohesion is lost in three stages during yielding, in response to changing increases in friction (Intermediate a, b and c in Figure 5.2.16). The peak cohesion is a combination of base mineral friction coefficient and peak mineral UCS strength, while the residual cohesion is a combination of the static gouge and residual axial strength.

Cohesion loss is a small-strain process, while confinement dependent strength arising from friction requires larger strain (Diederichs, 1999). Using this rationale, the cohesion is lost before the mineral has changed to gouge phase (Intermediate a in Figure 5.2.17). Cohesion loss can be also prescribed directly, with axial strength as a function of changing cohesion and friction coefficient up to residual cohesion (Intermediate a in Figure 5.2.17). In this case, residual axial strength is calculated by a combination of the static gouge and residual cohesion. For simplicity, strain at which the phase change to mineral gouge occurs is taken as 10x the strain at which cohesion reaches residual.
Figure 5.2.16: Schematic of strength property (cohesion or friction coefficient, $\mu$) and resulting axial strength as it changes with strain, where $\varepsilon_E$ and $\varepsilon_P$ are accumulated elastic and plastic strain, respectively. Peak strength parameters are used until the element fails and begins to accumulate plastic strain, at which three levels of intermediate strength parameters are used (a,b,c), as a function of increasing plastic strain, until the residual strength parameters are reached.

Figure 5.2.17: Schematic of strength property (cohesion or friction coefficient, $\mu$) and resulting axial strength as it changes with strain, where $\varepsilon_E$ and $\varepsilon_P$ are accumulated elastic and plastic strain, respectively. Peak strength parameters are used until the element fails and begins to accumulate plastic strain, at which two levels of intermediate strength parameters are used (a,b), as a function of increasing plastic strain, until the residual strength parameters are reached.
The latter method is used in this research, in which the cohesion drops to a residual value 1/20 of the peak value. The cohesion loss as a function of strain for input into the FLAC strain softening constitutive model is determined by estimating the plastic strain at which residual cohesion is reached, using the following relationship for model stability:

\[ \varepsilon_{\text{p, residual}} = \frac{\text{cohesion\_peak}}{\text{shear\_modulus}} \] 5.2.9

The strain rate used for modelling, is therefore different for each mineral, as well as for maximum and minimum values for the same mineral. Equation 5.2.9 defines the relationship between the cohesive strength of the material and the shear stiffness, and expresses the resistance to shearing through an intact mineral lattice in terms of bond strength and bond stiffness. This ratio gives an estimate of the strain required to overcome the shearing resistance. The tensile strength to strain relationship is not so easily determined, but for numerical stability and simplicity, it is associated with the cohesion loss strain rate. The strain level at which monomineralic gouge is expected to develop is one order of magnitude higher than the strain at which cohesion is lost.

The cohesion for mica should be equally anisotropic as the tensile strength and stiffness moduli. The UCS values in Table 5.2.3 do not specify whether or not they are parallel or perpendicular to the basal plane, but judging by the Young’s modulus values quoted along with the UCS values, they appear to have been tested parallel to the basal plane. Taking the same UCS to tensile strength ratio as for quartz and feldspar, all of which reflect failure through covalent bonds, the UCS perpendicular to the basal plane could be as high as 390 MPa. For the purposes of this research, this will be used as the UCS strength estimate perpendicular to the basal plane, and perpendicular peak cohesion is calculated based on this value.

5.2.3.6 Grain Boundaries

Grain boundaries are a special system within a mineral grain in that they can act as both crack initiators and crack arrestors. Their low stiffness (Diederichs, 1999), due to lattice misalignment (Figure 5.2.18) between similar mineral types or incompatible lattices between different mineral types, allows them to arrest intragranular cracks. When oriented oblique to maximum stress direction or parallel to maximum stress direction within the stress field, their lower cohesion and tensile strength; allows them to act as crack initiators. Grain boundaries have been found to contain large concentrations of microfractures in undeformed samples (Moore and Lockner, 1995), contributing to lower strength (Nasseri et al., 2005; Nasseri et al., 2002).
The lower density (Tromans and Meech, 2002) leads to lower strength and stiffness. For this reason, the grain boundaries in the FLAC model are given separate, but related, input parameters to reflect these property differences. These properties depend on the two mineral types adjacent to the boundary, and follow these rules of thumb: stiffness is half the stiffness of the softer mineral, cohesion and tensile strength are 85% of the average strength between the two minerals, and friction is whichever friction coefficient is lower. The strength-strain relationships follow the relationship in Equation 5.2.9.

5.2.4 Parametric Analysis of Strength, Stiffness and Dilation Combinations

Models of simple 100x200 element meshes with a circular inclusion were used to investigate the relative impacts on fracture behaviour of changes in strength, stiffness and dilation within the inclusion, with respect to the exterior during uniaxial compression. Both the inclusion and matrix properties were homogeneous strain softening Mohr-Coulomb constitutive models, with the inclusion properties varied with respect to the matrix. To ensure visible results, the stiffness and strength of the inclusion were either halved or doubled with respect to the matrix. As a baseline, two end-member models were run: one in which the inclusion is the same as the matrix, and one in which the inclusion is a hole. The case with a similar inclusion resulted in a series of conjugate fracture sets, while the case with the hole resulted in axial tensile fractures propagating in the direction of applied stress away from the boundaries. These results are somewhat artificial since the boundaries are not confined, but the relative impacts of the strain differences are valuable. Figure 5.2.19 summarises the results of the four different strength
Figure 5.2.19: Summary of stiffness-strength relativity testing (top) with corresponding representative images of failure behaviour (bottom), where pink is shear failure, purple is tensile failure, and vertical stress is contoured (blue=low, yellow=high).
and stiffness combinations. The images are also used to ensure that tensile processes function as expected in the FLAC model with this configuration. The tests were repeated with 25% and 50% dilation, but the results were not considerably different.

The rock types dealt with in this research are comprised mainly of quartz and feldspar, with various combinations of micas, chlorite, amphibole, mafic minerals, oxides and sulphides. The accessory minerals typically fall in the soft-weak, stiff-strong or simply stiff categories with respect to quartz and feldspar. The soft minerals will propagate fractures through the matrix either as a continuation of fractures in the inclusions (weak) or by inducing fractures (strong) by shedding stress into the matrix. Stiff minerals will localise stress and either fail early (weak) and essentially become soft-weak or will fail later than the matrix (stiff only or stiff-strong). These fracture behaviours have been observed in real rocks by several authors (Li et al, 2003; Li, 2001; Tapponier and Brace, 1976; Wong, 1982) as discussed in Chapter 4.2.2.

### 5.2.5 Verification of Constitutive Model

#### 5.2.5.1 Laboratory Testing Dataset

A dataset of UCS, triaxial and Brazilian strength tests (results summarised in Table 5.2.7 based on Roclab and available data) and samples from Stanstead granodiorite (courtesy: J. Archibald) was used as a baseline with which to verify the mineral-specific constitutive models as well as the texture creation algorithm. This granodiorite is composed of 70% feldspar (potassium feldspar and plagioclase), 20% quartz, and 9% mica and 1% of other accessory minerals. The grain sizes range from medium to coarse, 2-16mm, individual grains are rounded, to slightly elongated, although they do not define an anisotropic fabric. The mineral type, grain size and grain orientation distribution are isotropic (Figure 5.2.20).

Table 5.2.7: Summary of laboratory testing data from Stanstead granodiorite (courtesy J. Archibald)

<table>
<thead>
<tr>
<th>UCS MPa</th>
<th>Cohesion MPa</th>
<th>Friction Angle ((\phi+i)) at (\sigma&lt;0.1UCS)</th>
<th>Coefficient of Friction</th>
<th>Brazilian Tensile Strength MPa</th>
<th>Poisson’s Ratio</th>
<th>Young’s Modulus GPa</th>
<th>Failure angle</th>
</tr>
</thead>
<tbody>
<tr>
<td>148</td>
<td>28-41</td>
<td>62.5-64°</td>
<td>1.48</td>
<td>6.5 +/- 0.8</td>
<td>0.19</td>
<td>35</td>
<td>73°</td>
</tr>
</tbody>
</table>
5.2.5.2 Numerical Modelling Calibration Results

The texture and mineral composition of the Stanstead granodiorite were created in the FLAC model (Figure 5.2.21) and used in both the UCS and Brazilian test models, using the texture algorithm described in Appendix E.3. A series of numerical uniaxial and biaxial tests, as well as Brazilian tests were conducted and compared to the averaged laboratory test data.

5.2.5.2.1 Brazilian Tensile Testing

Brazilian testing of cores has resulted in a Brazilian tensile strength of 6.5MPa (courtesy J. Archibald). This value is low with respect to the UCS strength and the rock type. Examination of thin sections has shown that 70% of the quartz grains contain intragranular and transgranular microfractures extending only within adjacent quartz grains (as in Figure 5.2.13). This was also observed by Nasseri et al (2002) in granitoids and led to decreased fracture toughness values since pre-existing microcracks can be used to link new induced microcracks and eventually lead to rupture (Moore and Lockner, 1995). Results from numerical Brazilian tensile testing using high end tensile strength values from Table 5.2.7 give a Brazilian tensile strength of 16 MPa. By decreasing the input tensile strength for quartz to 2 MPa the Brazilian tensile strength of the modelled granodiorite is approximately 8.5-9.2 MPa, while 15% dilation increases this strength to approximately 9.3-11.3 MPa, which is slightly higher than the physical test strength and within three standard deviations of the dataset.

Figure 5.2.20: Photo and photomicrograph of Stanstead granodiorite showing isotropic nature of the material. Q=quartz, P=feldspar, predominantly plagioclase, M=mica, predominantly biotite.
Figure 5.3.21: Images of modelled Stanstead granodiorite in FLAC. Colours relate to mineral type as follows: turquoise = feldspar, green = quartz, red = mica, yellow = grain boundaries.

A comparison of the input tensile strength values required to obtain a fit to the Brazilian tensile strength of the Stanstead granodiorite reveals:

1. The input tensile strength is much higher than either the Brazilian tensile strength or the tensile failure stress at which the elements fail. A back-calculation of the stress intensity factor based on direct tensile strength test results gives values similar to quoted mineral-specific values, but relating the tensile strength to stress intensity factor values quoted for quartz suggest unrealistic initial crack lengths (Diederichs, 1999). This suggests that Brazilian tensile strength cannot be directly estimated based on the mineral-specific input tensile strengths, as the estimates would be unrepresentatively high.
2. Damage to even a small percentage of the minerals (70% of quartz, which comprise only 20% of the composite) can greatly lower the Brazilian tensile strength, demonstrating that pre-existing damage, either due to sampling or tectonic history, causing intragranular microfractures at the grain scale is important for Brazilian tensile strength and should be taken into account when characterising rock for excavation boundary yielding behaviour prediction.

In addition to the input tensile strength, the heterogeneity in stiffness moduli arising from the different minerals and their grain boundaries is critical to obtaining a fit to the Brazilian tensile strength of the Stanstead granodiorite, as seen in laboratory testing of Westerly granite (Tapponier and Brace, 1976; Wong, 1982). Tests run with homogeneous stiffness moduli resulted in Brazilian tensile strengths of 44 MPa and 60 MPa for low and average input tensile strengths, respectively. These values are clearly too high, and no amount of lowering the quartz tensile strength and cohesion could reduce the Brazilian tensile strength to fit with the Brazilian tensile strength of Stanstead granodiorite. In addition to the strength difference, the yielding behaviour is also different. If homogeneous stiffness moduli are used, the failure is distributed throughout the sample, while with heterogenous stiffness moduli, the failure is constrained within discrete yielding planes, interpreted as macrofractures (Figure 5.2.22).

![Figure 5.2.22: Locations of element failure leading up to composite sample failure for samples with homogeneous (left) and heterogeneous (right) stiffness moduli; all other input parameters are identical.](image-url)
5.2.5.2.2 UCS and Biaxial Testing

The goal of the UCS and biaxial model testing was to duplicate the UCS and triaxial values obtained from laboratory testing of the Stanstead granodiorite (courtesy J. Archibald). Based on the laboratory triaxial data (Figure 5.2.23), linear curve fitting of the Mohr circles joining \( \sigma_3 - \sigma_1 \) data the cohesion and friction angle were estimated for two instantaneous confining stress values: 5 MPa and 10 MPa. The fit lines give friction angles, \( \phi \), of 64° and 62.5°, and friction coefficients of 2.05 and 1.92, respectively. These are higher than the relationship demonstrated by Byerlee (1978), and much higher than the highest coefficient of friction of any of the individual minerals and mineral gouges. The shear stress intercepts gave cohesion values of 28 MPa and 41 MPa, respectively. These values are excessively high (for friction) and low (for cohesion) due to the limited range of data points restricted to low confinement \( \sigma_3 < 0.1 \sigma_1 \).

The modelled biaxial tests, modelled using average strength parameters and low tensile strength, suggests that the friction angle is approximately 30.5° and the coefficient of friction is 0.58, and a cohesion value of 48 MPa based on \( \sigma_1 \) vs \( \sigma_3 \) stress data from the tests (Figure 5.2.24) at a low instantaneous confining stress value. This shows that the model has a coefficient of friction that is slightly higher than the average initial coefficients of friction of the individual minerals. In order to explore the cause of the low friction, an instantaneous (\( \varepsilon = 1 \times 10^{-8} \), for model stability) friction gain of 45° caused by intragranular microfracture tortuosity is applied. The resulting biaxial strengths are plotted in Figure 5.2.24 and show that the composite rock friction angle is increased to 33.5° and the coefficient of friction is 0.66, with a cohesion value of 47.5 MPa. The cohesion values are higher than the physical lab test results with correspondingly lower coefficients of friction, even at low confinement \( \sigma_3 < 0.1 \sigma_1 \), suggesting that the model is not capable of simulating the confinement dependent fracture process. The model has been shown to be capable of simulating confinement independent failure, which is the confinement zone of interest to this research. The Mohr-Coulomb parameters at low instantaneous minor principal stresses are shown for the laboratory and model data in Tables 5.2.8 and 5.2.9, respectively.
Figure 5.2.23: Mohr-Coulomb graph of triaxial test $\sigma_1$ and $\sigma_3$ data for Stanstead granodiorite data (courtesy of J. Archibald) with Mohr circles and estimated Mohr-Coulomb parameter fits.

Figure 5.2.24: Mohr-Coulomb graph of biaxial test $\sigma_1$ and $\sigma_3$ data for modelled Stanstead granodiorite with delayed friction (dashed lines) and with instantaneous 45° friction (solid line) with Mohr circles and estimated Mohr-Coulomb parameter fits.
Table 5.2.8: Summary of Mohr-Coulomb parameters derived from biaxial testing of Stanstead granodiorite (data courtesy of J. Archibald) taken at 4 different instantaneous minor principal stress values

<table>
<thead>
<tr>
<th>Instantaneous $\sigma_3$ MPa</th>
<th>Cohesion MPa</th>
<th>Friction Angle</th>
<th>Coefficient of Friction</th>
</tr>
</thead>
<tbody>
<tr>
<td>5</td>
<td>15</td>
<td>64</td>
<td>2.05</td>
</tr>
<tr>
<td>10</td>
<td>16</td>
<td>62.5</td>
<td>1.92</td>
</tr>
</tbody>
</table>

Table 5.2.9: Summary of Mohr-Coulomb parameters derived from biaxial testing of Stanstead numerical models with delayed friction and instantaneous friction gain to 45°, taken at 4 different instantaneous minor principal stress values

<table>
<thead>
<tr>
<th>Instantaneous $\sigma_3$ MPa</th>
<th>Delayed Friction</th>
<th>Immediate Friction 45°</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Cohesion MPa</td>
<td>Friction Angle</td>
</tr>
<tr>
<td>5</td>
<td>48</td>
<td>30.5</td>
</tr>
<tr>
<td></td>
<td>47.5</td>
<td>33.5</td>
</tr>
</tbody>
</table>

The instantaneous friction gain to 45° is clearly an improvement on the numerical model results. The principle behind this increase is the need to explicitly account for friction arising from the tortuosity of the newly created fractures inside the mineral grains, since FLAC does not create discrete fractures when an element (or grain) fails. This is necessary to obtain more realistic UCS and biaxial model test results, for which the friction component of strength is important due to contact at the fracture surfaces from the (potentially minimal, but nevertheless present) confinement. At under low confinement conditions, as in excavation boundaries, however, this friction effect is not as critical, as shown by in Figure 5.2.25, where extension cracks propagate freely, without contact of the fracture surfaces, in which case the frictional component of strength is not mobilised.

The upper envelope (applicable at higher confinement) is analogous to the lab values with instantaneous, while the lower envelope (applicable at low confinement) is analogous to the model values with delayed friction gain, as shown in Figure 5.2.26. The constitutive model used for UCS and Brazilian tensile modelling follows the schematic in Figure 5.2.27, and generates the most realistic results. The value of 45° was selected based on the need to obtain a considerable change in results between the delayed friction gain and instantaneous friction gain, although the true value is not currently known, and because of time considerations was not tested. The friction is applied at $10^{-8}$ plastic strain for model stability reasons, and is, therefore not perfectly instantaneous, but is five orders of magnitude smaller than the strain at which cohesion is lost, making it essentially instantaneous in this application.

This analysis shows that the constitutive model cannot capture the shearing of minerals that occurs during triaxial failure at high confining stress. A similar phenomenon was observed by Diederichs (1999) in which the peak strength envelope slopes for models of polycrystalline
Figure 5.2.25: Schematic normalised principal stress graph showing strength envelope drop at low confinement (after Diederichs, 2003)

Figure 5.2.26: Peak strength envelopes for Stanstead laboratory test data (courtesy of J. Archibald), Stanstead model data with 15% dilation and Stanstead model data with 15% dilation and instantaneous friction increase.
Figure 5.2.27: Schematic of strength property (cohesion or friction coefficient, \( \mu \)) and resulting axial strength as it changes with strain, where \( \varepsilon_E \) and \( \varepsilon_P \) are accumulated elastic and plastic strain, respectively. Peak strength parameters are used until the element fails and begins to accumulate plastic strain, at which friction increases instantaneously, followed by two levels of intermediate strength parameters (a,b), as a function of increasing plastic strain, until the residual strength parameters are reached.

rocks data were much lower than for laboratory test values. The slopes for data points from the crack interaction stress (determined directly in the models) and the axial stress-strain graph nonlinearity point were found to have similar slopes in both model and laboratory test data of granite (Diederichs, 1999). It was also found that the peak strength envelopes for laboratory samples loaded very slowly resulted in a much lower slope, approximately 3.8, compared to quickly loaded samples, whose peak strength envelope slope was approximately 7.5.

This suggests that peak values in the biaxial numerical model tests may actually be more representative of the crack initiation threshold, rather than true laboratory peak strength. This phenomenon is particularly important for biaxial model test results since the impact of confining stress is not correctly modelled, as demonstrated by the lower slope angles of modelled test results. With respect to UCS testing, however, the confining stress is not an issue as the test is
undertaken in the unconfined zone of Figure 5.2.26, and the behaviour of the model test can be taken as an analogue to the laboratory test. Issues arising from 2-dimensional versus 3-dimensional samples for UCS testing are independent of this phenomenon. This must be taken into consideration when interpreting the unconfined and confined model data presented in this research since finding a solution to the biaxial test does not fall within the scope of this research.

### 5.2.5.3 Summary of Fracture Behaviour

#### 5.2.5.3.1 Compressive Failure

The mineral-specific behaviour described in the literature (Li et al, 2003; Li, 2001; Tapponier and Brace, 1976; Wong, 1982) and discussed in Sections 4.2.2.2 and 5.2.4 was also observed in the 2-D FLAC UCS models. The micas, being soft in shear (Figure 5.2.29, centre right), induce failure in the surrounding stiffer, albeit stronger, feldspar (Figure 5.2.29, centre left), as discussed in Li (2001). A tensile fracture propagating (but ultimately abandoned) through feldspar (Figure 5.2.29, lower right, circled in white) can be halted at the more soft and slightly weaker grain boundary (Figure 5.2.29, lower left, red circle), as shown by Martin (1994) and Li (2001). A tensile fracture propagating through feldspar (Figure 5.2.31, right showing strain due to tensile fracturing) can be offset by a more compliant mica (Figure 5.2.30, left showing deviation of fracture), which does not propagate the fractures well at a large angle to the cleavage, as in Tapponnier and Brace (1976) and Li (2001). A macrofracture will eventually develop in uniaxial loading, at which point the fracture path will include all mineral types (Figure 5.2.31). In this series of images, the quartz is microfractured, and is therefore weaker in tension and more compliant than unfractured quartz, and is seen as containing numerous fractures in the FLAC images.

<table>
<thead>
<tr>
<th>Mineralogy</th>
<th>Failure</th>
</tr>
</thead>
<tbody>
<tr>
<td>Feldspar</td>
<td>Previous Failure, no longer active</td>
</tr>
<tr>
<td>Grain Boundaries</td>
<td>Active Tensile Failure</td>
</tr>
<tr>
<td>Quartz</td>
<td>Active Shear Failure</td>
</tr>
<tr>
<td>Biotite</td>
<td>Previous Failure along ubiquitous joint, no longer active</td>
</tr>
<tr>
<td></td>
<td>Active Tensile Failure along ubiquitous joint</td>
</tr>
<tr>
<td></td>
<td>Active Shear Failure along ubiquitous joint</td>
</tr>
</tbody>
</table>

Figure 5.2.28: Legend for Figures 5.2.30-5.2.33.
Figure 5.2.29: FLAC output of failed modelled Stanstead Granodiorite showing induced failure around biotite (left) and the shear strain intensity of the corresponding region (right; yellow-green is higher strain); Red (left) and white (right) circles highlighting tensile fracture propagating through feldspar; Legend in Figure 5.2.29.

Figure 5.2.30: FLAC output of failed modelled Stanstead Granodiorite showing induced failure tensile failure propagating through feldspar, then moving around biotite (left) and the shear strain intensity of the corresponding region (right; yellow-green is higher strain); Legend in Figure 5.2.29.
5.2.5.3.2 Tensile Failure

Tensile failure is less dependent on fracture accumulation and coalescence than the UCS failure in FLAC, since the tensile stress is generated at the centre of the sample and leads to tensile failure in the material of least resistance within this zone of tensile stress. Figure 5.2.32 shows the centre of a failed Brazilian model in which the failure began in a mica grain, and propagated up and down to form a nearly linear fracture surface. In this case, very little deviation by other minerals occurs, and the fracture propagates nearly unhindered. This is similar to the failure surfaces observed in laboratory indirect tensile tests, such as Brazilian and point load tests (Figure 5.2.33).
Figure 5.2.32: FLAC output of failed modelled Stanstead Granodiorite in Brazilian test showing induced macro fracture propagating through feldspar, quartz and biotite (left) and the shear strain intensity of the corresponding region (right; yellow-green is higher strain); Legend in Figure 5.2.26.

Figure 5.2.33: Samples diametral (left) and axially (right) fractured by point load testing, showing nearly linear fracture surfaces induced by tensile stress concentrations.

### 5.2.5.3.3 UCS and Acoustic Emissions

A series of ten UCS tests with acoustic emissions were undertaken on samples collected from the Gotthard tunnel (See Appendix B.3). Thin sections of the tested rocks were cut and characterised according to the methodology developed in Section 4.2. From these data, mineralogy, grain size and fabric relating to each individual sample were input into the FLAC model, which was then used to simulate a 2-D UCS test. The initiation, coalescence and peak stresses as well as the elastic stiffness from each test were compared, according to the
Table 5.2.10: Mean and standard deviation values of difference between modelled and physical laboratory peak, initiation and coalescence values for eight tests.

<table>
<thead>
<tr>
<th>Value</th>
<th>Difference Mean</th>
<th>Difference Standard Deviation</th>
</tr>
</thead>
<tbody>
<tr>
<td>Peak</td>
<td>0.18</td>
<td>0.11</td>
</tr>
<tr>
<td>Initiation Low</td>
<td>0.15</td>
<td>0.15</td>
</tr>
<tr>
<td>Initiation High</td>
<td>0.2</td>
<td>0.18</td>
</tr>
<tr>
<td>Coalescence</td>
<td>0.18</td>
<td>0.09</td>
</tr>
</tbody>
</table>


The peak values have a mean difference of nearly 18%, and the initiation and coalescence have mean difference of 15% and 18%, respectively (Table 5.2.10), showing that the modelled results tend to be within 20% of the physical laboratory UCS results. In general, the results have quite good fits, and show that the model is capable of simulating realistic UCS behaviour for samples with different mineral contents, grain sizes and fabrics.

The acoustic emissions for the numerically modelled samples does not produce AE-stress graphs as outlined in Diederichs et al. (2004) since crack closure is not simulated in the models, thus no transition to increased strength at high confinement. This makes interpretation of the initiation and coalescence thresholds based on AE data inconclusive when compared to the estimates obtained using the stress-strain graphs. For the modelled data, therefore, the thresholds are interpreted using the stress-strain graphs.

5.2.5.3.4 Discussion of Modelled versus Physical Results

The UCS and biaxial values are lower in the modelled tests than in the physical tests, in part due to the limitations of simulating a three-dimensional test in 2-D. The hoop stresses that develop in physical UCS models (Diederichs, 2003; Diederichs et al., 2004) and likely contribute to higher UCS strength are not developed in the 2-D models. The Brazilian tensile strengths are higher in the modelled tests than in the physical tests, partly due to the continuum nature of the model in which actual breaks in the rock are not developed, in contrast to the physical models in which tensile fractures tear the sample apart with considerable speed and energy, likely contributing to decreased physical strength values.

In general, the best fits to UCS, biaxial and Brazilian tensile data require high tensile strength of intact minerals, and low tensile strength and stiffness of fractured minerals. The fits are not perfect, and more work is required to ensure that the models can fit all three tests, particularly the triaxial test at high confining stress. The fits obtained are sufficient for the
purposes of this research, in light of the fracture behaviour described in Section 5.2.5.3, which mimic behaviour described from physical tests. The goal of this research is to determine the relative impacts of different geological parameters on chipping and tunnel face stability, and the results obtained from the calibration suggest that the input parameters are able to fulfill this task.

Improvements to modelled UCS strength could be accomplished by better determining the friction angle and cohesion, and as well as varying the strain rate at which residual friction is mobilised. In particular, increasing the friction angle nearly instantaneously (strain of $1 \times 10^{-8}$) increased the calculated rock friction angle by about 5% with respect to the physical laboratory results from triaxial testing. Further improvements with respect to friction, either in terms of input parameters, mobilisation rate or further explicit mobilisation of macro friction to obtain model biaxial results that are comparable to laboratory triaxial results could be undertaken.

Improvements to Young’s modulus could be made by further varying stiffness of the minerals, or their grain boundaries, while improvements to Brazilian tensile strength could be made by decreasing input tensile strength, but this must be verified with UCS to ensure it does not cause a deviation.

### 5.2.6 Summary of Selected Input Parameters

The results of the discussion presented in Section 5.2 are summarised in Table 5.2.11. The ranges of values represent the following:

- **Cohesion**: peak and residual following the values outlined in Section 5.2.3.5 and Figure 5.2.34.

- **Tensile strength**: The high-end values from Figures 5.2.8 to 5.2.10 result in the best fit for Brazilian and UCS tests calibrated to the Stanstead granodiorite, with instantaneous drop to 0 as shown in Figure 5.2.34.

- **Mineral friction coefficients**: base friction coefficient relates to friction of polished mineral surfaces and asperities, as calculated in Section 5.2.3.3.

- ‘Instantaneous’ friction gain to 45° at $\varepsilon = 1 \times 10^{-8}$ in all mineral matrix constitutive models, as shown in Figure 5.2.34.

- ‘Instantaneous’ friction gain to 26° at $\varepsilon = 1 \times 10^{-8}$ in mica ubiquitous joint constitutive models to maintain numerical stability.

- **Gouge**: monomineralic gouge friction coefficients, as shown in Figure 5.2.34; note that the gouge coefficient of friction for biotite is calculated based on the ratio of gouge to polished mineral surface for muscovite.
Elastic Moduli: low and high are second standard deviation of the normal distribution for quartz and feldspar.

Mohr-Coulomb plots representing the average strength envelopes of quartz, feldspar and mica corresponding peak and residual strength in the constitutive model presented in Figure 5.2.34 are shown in Figures 5.2.35-5.2.36.

Figure 5.2.34: Schematic of strength property (cohesion or friction coefficient, $\mu$) and resulting axial strength as it changes with strain, where $\varepsilon_E$ and $\varepsilon_P$ are accumulated elastic and plastic strain, respectively. Peak strength parameters are used until the element fails and begins to accumulate plastic strain, at which friction increases instantaneously, followed by two levels of intermediate strength parameters (a,b), as a function of increasing plastic strain, until the residual strength parameters are reached.
Table 5.2.11: Summary of selected FLAC input strength parameters for rock modelling

<table>
<thead>
<tr>
<th>Mineral</th>
<th>Cohesion MPa</th>
<th>$\sigma_t$ MPa</th>
<th>Friction Coefficients</th>
<th>Poisson Ratio</th>
<th>Elastic Moduli GPa</th>
<th>Strain $\varepsilon \times 10^{-3}$</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Peak</td>
<td>Residual</td>
<td>Base</td>
<td>Gouge</td>
<td>Young’s</td>
<td>Shear</td>
</tr>
<tr>
<td>Quartz</td>
<td>109</td>
<td>5</td>
<td>34</td>
<td>0.132</td>
<td>0.65</td>
<td>0.19</td>
</tr>
<tr>
<td>Feldspar</td>
<td>80 – 125</td>
<td>4 – 6</td>
<td>36</td>
<td>0.12 – 0.59</td>
<td>0.8</td>
<td>0.28</td>
</tr>
<tr>
<td>Biotite parallel</td>
<td>48</td>
<td>2</td>
<td>4.5</td>
<td>0.31</td>
<td>0.43</td>
<td>0.18</td>
</tr>
<tr>
<td>Biotite perpendicular</td>
<td>95</td>
<td>5</td>
<td>39</td>
<td>0.78</td>
<td>0.43</td>
<td>0.053</td>
</tr>
</tbody>
</table>
Figure 5.2.35: Peak Mohr-Coulomb strength envelope for quartz, feldspar and mica using average values from Table 5.2.11, as described in Section 5.2.3.4.

Figure 5.2.36: Residual Mohr-Coulomb strength envelope for quartz, feldspar and mica using average values from Table 5.2.1, as described in Section 5.2.3.4.
5.3 Parametric Analysis of F Factors

5.3.1 Introduction

The generated textures demonstrated in Appendix E.3 were used to calibrate the Geomechanical Characterisation scheme developed in Chapter 4.2 through a parametric analysis of the F-Factors in two geometrically simple two-dimensional models: a Brazilian tensile test and a uniaxial compression test (see Appendix E.2). The modelled strength tests were first conducted on an initial standard case (Stanstead Granodiorite from Appendix E.3) from which the F-Factors were systematically varied and the resulting changes were monitored. The models do not provide explicit values for the F-Factors, but rather they were used to weigh the F-Factor categories according to their relative impacts on the following monitored parameters and model outputs:

- Stress- axial strain curve
- Peak strength
- Ratios of failed minerals
- Impact of grain boundaries
- Dominant fracture method (tensile, shear, or a combination of the two)
- Resulting failure shape, or failure angle
- Damage initiation, interaction and peak thresholds

Based on the eventual failure characteristics of the tests (see Appendix E.4), as well as comparison between Brazilian tensile strength and UCS for samples with the same F-Factors, each sample was categorised into spalling (tensile dominated) and non-spalling (shear dominated) sensitive. These categorisations were used to investigate the impact of each F-Factor on the spalling sensitivity of modelled rocks, in a similar fashion as for Section 4.3.

5.3.2 Parametric Investigation of F-Factors

The sensitivity of the modelled rocks in the Brazilian tensile test and UCS test to each of the F-Factors was investigated by varying only one single factor at a time and comparing the results. In some cases, the dependence of a test on two parameters was established by testing the sensitivity to a particular parameter in models in which other parameters were varied.

The sensitivity of the modelled rocks to mineralogy was modelled by a series of three tests in which the mica content was varied, while keeping the quartz to feldspar ratio constant, a
series of four tests in which the quartz content was varied while keeping the mica to feldspar ratio constant, and a series of three tests in which the quartz to feldspar ratio was varied while keeping the mica content constant.

The sensitivity of the modelled rocks to grain size was modelled using two methods: rocks without grain boundaries and rocks with grain boundaries. The rocks with 0.5mm grain size were below the resolution of the element size in the model. This element size was selected as the optimal size for the purposes of this research (see Appendix E.2). In order to derive a relative comparison for rocks with different grain sizes, a series of three tests with grain sizes from 0.5mm to 8mm without grain boundaries, and an overlapping series of four tests with grain sizes from 4mm to 16mm were performed. This also allows a comparison of the effect of grain boundaries on test results.

The sensitivity of the modelled rocks to different fabric types and intensities was modelled by applying the fabric generation algorithm discussed in Appendix E.3. The eight modelled fabrics were selected according to the divisions discussed in Section 4.2 and the models were created using thin section images of real rocks as a template (as discussed in Appendix E.3). The mineralogy and grain size are related to the fabric type and its intensity, and they are therefore not held constant with respect to changing fabric type. In the case of the fabric modelling the mineralogy and grain size change with fabric, but the impact of each is examined, categorised by fabric type and intensity. All fabric types are aligned at 30° to the loading axis, as this is the angle at which UCS is most impacted by the fabric (Hoek, 2007). This angle was selected to ensure the greatest impact from fabric such that the test results could be better interpreted.

5.3.2.1 Brazilian Tensile Strength Test Modelling

Three mineralogy and grain size and eight fabric tests were modelled for each F-Factor combination. The total number of tests was 112, the results of which are contained in Table 5.3.1, although only one set of outputs for each combination was selected for Appendix E.4. These results were analysed and used to interpret the impacts of each parameter on the Brazilian tensile test results and the sample failure process, as shown in the following section.
Table 5.3.1: Summary of key output parameters and results from parametric Brazilian tensile strength (MPa) numerical modelling, with grain size in mm.

<table>
<thead>
<tr>
<th>Sample</th>
<th>Brazilian Tensile</th>
<th>Mineralogy</th>
<th>Quartz to Feldspar</th>
<th>Grain Size</th>
<th>Fabric Angle</th>
</tr>
</thead>
<tbody>
<tr>
<td>m1</td>
<td>17.3</td>
<td>6</td>
<td>20</td>
<td>74</td>
<td>0.27</td>
</tr>
<tr>
<td>m1</td>
<td>19.0</td>
<td>4</td>
<td>22</td>
<td>74</td>
<td>0.30</td>
</tr>
<tr>
<td>m1</td>
<td>20.4</td>
<td>5</td>
<td>20</td>
<td>75</td>
<td>0.27</td>
</tr>
<tr>
<td>m2</td>
<td>15.3</td>
<td>18</td>
<td>17</td>
<td>65</td>
<td>0.26</td>
</tr>
<tr>
<td>m2</td>
<td>16.7</td>
<td>16</td>
<td>16</td>
<td>68</td>
<td>0.24</td>
</tr>
<tr>
<td>m2</td>
<td>16.1</td>
<td>16</td>
<td>15</td>
<td>69</td>
<td>0.22</td>
</tr>
<tr>
<td>m3</td>
<td>14.5</td>
<td>25</td>
<td>17</td>
<td>58</td>
<td>0.29</td>
</tr>
<tr>
<td>m3</td>
<td>15.3</td>
<td>23</td>
<td>16</td>
<td>61</td>
<td>0.26</td>
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<tr>
<td>m3</td>
<td>15.4</td>
<td>23</td>
<td>16</td>
<td>61</td>
<td>0.26</td>
</tr>
<tr>
<td>q1</td>
<td>14.8</td>
<td>10</td>
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</tr>
<tr>
<td>q1</td>
<td>17.6</td>
<td>9</td>
<td>41</td>
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<td>0.82</td>
</tr>
<tr>
<td>q1</td>
<td>15.3</td>
<td>10</td>
<td>40</td>
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</tr>
<tr>
<td>q2</td>
<td>20.9</td>
<td>6</td>
<td>70</td>
<td>24</td>
<td>2.92</td>
</tr>
<tr>
<td>q2</td>
<td>18.1</td>
<td>5</td>
<td>70</td>
<td>25</td>
<td>2.80</td>
</tr>
<tr>
<td>q2</td>
<td>21.5</td>
<td>6</td>
<td>69</td>
<td>25</td>
<td>2.76</td>
</tr>
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<td>20.3</td>
<td>3</td>
<td>84</td>
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</tr>
<tr>
<td>q3</td>
<td>23.9</td>
<td>3</td>
<td>85</td>
<td>12</td>
<td>7.08</td>
</tr>
<tr>
<td>q3</td>
<td>21.4</td>
<td>3</td>
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<td>q35</td>
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<tr>
<td>q35</td>
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<td>3</td>
<td>91</td>
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<td>15.17</td>
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<td>22.8</td>
<td>4</td>
<td>91</td>
<td>5</td>
<td>18.20</td>
</tr>
<tr>
<td>q4</td>
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<td>47</td>
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<tr>
<td>q5</td>
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<td>12</td>
<td>41</td>
<td>47</td>
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5.3.2.1.1 Mineralogy Sensitivity

Several graphs were generated comparing mica content to Brazilian tensile strength in which the data were categorised by their variation in a third input parameter, which could be grain size, quartz to feldspar ratio or fabric type and intensity. The graphs were examined to determine the relationship between mica content and Brazilian tensile strength, as well as to determine the impact of this relationship on the other parameters. This is particularly important for the fabric type and intensity since the mica content will vary according to these characteristics.

Figure 5.3.1 contains results from all tests for which the mineralogy was varied, where Brazilian tensile strength is compared to the mica content. The Brazilian tensile strength decreases according to a power law with increasing mica content, regardless of the quartz to feldspar ratio, with an $R^2$ value of 0.7. This negative relationship is similar to the relationship with PLT index strength for Southern Aar granite in Section 3.3.6.1. This is further shown in Figures 5.3.2 and 5.3.3 in which the Brazilian tensile strength is compared to quartz to feldspar ratio with low mica content less than 10%, as well as with constant mica at around 10%. The results for the constant mica content and varying quartz to feldspar ratio do not define any relationship with Brazilian tensile strength, while the results with low mica content and varying quartz to feldspar ratio show a power relationship with Brazilian tensile strength. This suggests that mica is an important contributor to tensile strength, and when more than 10% is present, it is capable of masking the effect of quartz to feldspar ratio on Brazilian tensile strength. A positive relationship with quartz to feldspar ratio has been observed in laboratory UCS testing of granitic rocks from Turkey (Tugrul and Zarif, 1999), while a negative relationship was found with quartz content and Brazilian tensile strength of granitic rocks from California, but this was attributed to the differences in quartz shape and interlocking, rather than the actual quartz content (Merriam et al., 1970). The relationship in Figure 5.3.2 shows that the impact of increasing quartz to feldspar ratio levels out at higher ratios, suggesting that increasing quartz to feldspar ratio will cease to increase Brazilian tensile strength beyond a certain ratio.
Figure 5.3.1: Brazilian tensile strength versus mica content categorised according to quartz to feldspar ratio.

Figure 5.3.2: Brazilian tensile strength versus quartz to feldspar ratio categorised according to mica content < 8%.
Figures 5.3.4 and 5.3.5 contain comparisons of mica content to Brazilian tensile strength, categorised by fabric type and intensity, aligned parallel and perpendicular to the loading axis, respectively. The graph in Figure 5.3.4 shows a general decrease in Brazilian tensile strength with increasing mica content, which is also related to fabric intensity. The relationship within individual fabric types, however, is variable, suggesting that while mica plays a role in the Brazilian tensile strength of rocks with fabric, it is not the only parameter that plays such a role. The graph in Figure 5.3.5 does not show any relationship between mica content and Brazilian tensile strength in general, and minor relationships within the fabric types. This suggests that with fabric oriented perpendicular to the loading axis, the mica content does not play a major role in the Brazilian tensile strength.
Figure 5.3.4: Brazilian tensile strength versus mica content categorised according to fabric type aligned parallel to the axis of loading.

Figure 5.3.5: Brazilian tensile strength versus mica content categorised according to fabric type aligned perpendicular to the axis of loading.
5.3.2.1.2 Grain Size Sensitivity

The data presented in Figure 5.3.6 suggest that grain size has a quadratic relationship with Brazilian tensile strength, regardless of whether or not the grains have grain boundaries, similar to the relationship with PLT index strength of Southern Aar granite in Section 3.3.6.1. The quadratic relationship exists in part due to a scale relationship: at the small grain size end fracture propagation is more difficult due to cracks being arrested at the numerous interfaces between grains (with or without grain boundaries). At the larger grain size end the strength is more dependent on the highest percentage fraction mineral, which in this series of tests is feldspar. The test approaches the tensile strength of feldspar (see Table 5.2.9), but the scatter in the points increases with increased grain size demonstrating the dependence of the test on the location of the grains with respect to the sample and the location of initiating tensile fractures. A large mica grain located in the sample centre can result in a lower test result, whereas micas on the exterior of the sample will have very little impact on the test result. This relationship, therefore, is dependent on the mineralogy, since a material with a higher percentage of micas as opposed to feldspars, for example, may not have a quadratic relationship due to test results approaching mica tensile strength at the large grain size end.

Figure 5.3.6: Brazilian tensile strength versus grain size categorised according to presence of grain boundaries.
5.3.2.1.3 Fabric Type and Intensity Sensitivity

Figures 5.3.7 and 5.3.8 contain a comparison between fabric type and intensity and the Brazilian tensile strength, with the fabric aligned parallel and perpendicular to the loading axis, respectively. The data in Figure 5.3.7 show a decrease in Brazilian tensile strength with fabric intensity for mineral preferred orientation, in contrast to the relationship with PLT index strength for SAG in Section 3.3.6.1, although this may be in part due to mica content according to the relationship shown in Figure 5.3.4. An examination of the images in Appendix E.4 show that widely spaced Mineral Preferred Orientation (MPO) requires failure through feldspars since the widely spaced micas are not located optimally in the sample to facilitate failure. The narrowly spaced MPO sample contains mica grains in the tensile zone and failure is a combination of failure of micas and feldspar bridges.

Samples with schistosity do not define a clear relationship between intensity and Brazilian tensile strength, similar to the relationship with PLT index strength for SAG in Section 3.3.6.1, although the results can be related to the failure progression arising from a combination of grain size, mica content and alignment. The images in Appendix E.4 show that the failure for domainal schistosity occurred predominantly as slip along the cleavage planes of the large, continuous mica that surround the large feldspar and quartz microlithons. The failure for the Type 1 schistosity is mostly tensile failure induced in the feldspar microlithons by the deforming micas. The failure in Type 2 schistosity is mostly in the closely spaced micas and induced through feldspar bridges.

The samples with cleavage define a negative relationship with Brazilian tensile strength, similar to the relationship with PLT index strength for SAG in Section 3.3.6.1, and their failure patterns demonstrate the relationship between the combination of grain size, mineralogy and alignment with Brazilian tensile strength. The images from Appendix E.4 show that for domainal and intermediate cleavage the small mica grains initiate the failure, but it occurs predominantly in the feldspar and quartz grains, as observed by Li (2001). The difference between domainal and intermediate cleavage arises from the greater mica content, and hence narrower spacing in the intermediate cleavage. Failure in the continuous cleavage occurs nearly entirely as slip along the cleavage planes of the aligned and connected feldspar grains.
Figure 5.3.7: Brazilian tensile strength categorised into fabric types and intensities from less to more intense left to right, for fabric aligned parallel to the axis of loading; the white points are data points, while the black points are averages within each fabric type.

These results suggest that for fabric aligned parallel to the loading axis, failure through the micas (domainal schistosity, Type 2 schistosity and continuous cleavage) is easier than inducing failure through feldspar bridges (Type 1 schistosity, narrow MPO, domainal and intermediate cleavage), which is easier than inducing failure nearly entirely through feldspar (widely spaced MPO) resulting in a lower Brazilian tensile strength.

The trends in Figure 5.3.8 are less defined than those in Figure 5.3.7 and may arise from the mica content, since it was shown with Figure 5.3.5 that mica content plays a minor role in Brazilian tensile strength in this fabric orientation. As shown in the failure images in Appendix E.4, the dominant failure type for all minerals is tensile failure, since the mica cleavage planes are not aligned favourably for slip. The failure in widely and narrowly spaced MPO is primarily through feldspar grains, and the mica content is likely the main contributor to Brazilian tensile strength, as shown in Figure 5.3.5. The failure patterns for the samples with schistosity show a decrease in the localisation of the failure with increase in intensity, where the larger feldspar microlithons in the domainal schistosity have through going tensile fractures. The Type 1 and Type 2 schistosity samples have numerous failed elements that did not coalesce into large tensile fractures, but provided opportunities for fractures to interact, and led to localisation.
The failure patterns for the samples with cleavage are very similar and arise from numerous failed elements. An examination of the progression of the failure, however, shows that at 95% of peak strength, the sample with intermediate cleavage is already showing localisation at several locations, while the sample with domainal cleavage has a single location of localisation. This suggests that the fracture in the domainal cleavage propagated from one location, while for intermediate cleavage it propagated from several locations at once. The failure in the sample with continuous cleavage also has several locations of high strain, but the strains are lower at 95%, suggesting that localisation has not yet fully developed. These differences likely arise as a combination of mineral spacing and grain size, however, the true relation cannot be determined from these data.

These results show that for fabric oriented perpendicular to the loading axis, the variation is less pronounced and the degree to which initiation, interaction and coalescence develop to finally localise into a single tensile fracture is most important. Contributing factors to rapid localisation include high numbers of microfractures (failed elements in the model), which increase the likelihood of interaction, and strength and stiffness difference between mineral types which promote failure in some minerals more than others (for example feldspar is difficult to promote failure through, as seen in the domainal schistosity and widely spaced MPO samples).

Figure 5.3.8: Brazilian tensile strength categorised into fabric types and intensities from less to more intense left to right, for fabric aligned perpendicular to the axis of loading; the white points are data points, while the black points are averages within each fabric type.
5.3.2.2 Uniaxial Compressive Test Modelling

The failure process of UCS testing, in which elements fail throughout the entire sample, and a large percentage of elements must fail before localisation occurs, renders the UCS test very time intensive in FLAC modelling. For this reason only three tests for each combination of F-Factors were completed. The results for 75 individual tests are presented in Table 5.3.2, and one set of outputs for each combination is represented in Appendix E.4. These results were manipulated and used to interpret the impacts of each parameter on the UCS test results and the sample failure process, as shown in the following section. The initiation and coalescence were interpreted from the axial and lateral stress-strain graphs.
Table 5.3.2: Summary of key output parameters and results from parametric UCS numerical modelling, strength in MPa, elastic modulus in GPa and grain size in mm.

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<th>Mineralogy</th>
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<th>Grain Size</th>
<th>Spall Sensitivity</th>
<th>Fabric Angle</th>
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5.3.2.2.1 Mineralogy Sensitivity

As with the Brazilian tensile strength results, the UCS has a power relationship with mica content, shown in Figure 5.3.9, similar to the negative relationship with PLT index strength for SAG in Section 3.3.6.1. The points with constant quartz to feldspar ratio define a relationship similar to the relationship for all points (with different grain size and quartz to feldspar ratio). The latter relationship shows considerable scatter in the data points, demonstrating that, while mica content is an important factor for UCS, other factors also contribute it. This has also been shown for physical laboratory biaxial testing of schists and gneisses with varied fabric types, intensities and mica contents (Shea and Kronenberg, 1993). The samples tested have up to 38% mica to investigate the impact of higher mica contents on UCS strength. This range is beyond the range of mineral compositions of igneous rocks (rocks without fabric) encountered in nature, for which mica contents are rarely higher than 10-15% (Figure 5.3.10). An overview of UCS test results for which mineral content is available (Tugrul and Zarif, 1999; Göransson et al., 2004; Prikryl, 2001) showed that most rocks tested had mica content between 2-12%, and those samples that had higher mica content were metamorphic rocks (Prikryl, 2001). The mica content in SAG is less than 15% for rocks without fabric, although it is as high as 65% for rocks with fabric. The data found in Prikryl (2001) shows a similar power relationship between mica content and UCS, for all points (including different grain size and quartz to feldspar content) with considerable scatter (Figure 5.3.11). It is possible that beyond a certain percentage (around 12%) increases in mica content do not drastically affect UCS strength.

Figure 5.3.12 shows that no relationship between mica content and UCS is discernable when samples with variable grain size are plotted. The scatter in this data shows that for this dataset mica content is not the most important factor. Figure 5.3.13 shows that in general a weakly negative relationship exists between mica content and UCS for samples with fabric. Within the fabric types, however, negative relationships exist between mica content and UCS for samples with schistosity and cleavage, and a weakly positive relationship exists for samples with mineral preferred orientation. These relationships do not differ greatly from the higher end mica content portion of the graphs in Figures 5.3.9 and 5.3.11.
Figure 5.3.9: UCS versus mica categorised according to quartz to feldspar ratio.

Figure 5.3.10: Schematic mineral composition chart for igneous rocks (modified from Plummer et al., 2003)
Figure 5.3.11: UCS versus mica categorised according to quartz to feldspar ratio or presence of fabric (data from Pikryl, 2001)

Figure 5.3.12: UCS versus mica categorised according to fabric type aligned at 30° to the axis of loading.
As for Brazilian tensile strength, the quartz to feldspar ratio has a positive relationship with UCS. The data in Figures 5.3.13 and 5.3.14 show a well defined relationship between quartz to feldspar ratio and UCS when the mica content is held constant, which has also been observed in physical UCS testing of granitic rocks from Turkey (Tugrul and Zarif, 1999) and from Sweden (Göransson, et al., 2004). Similar to Brazilian tensile strength, a negative relationship was found between quartz content and laboratory UCS of granitic rocks from California, but this was attributed to the differences in quartz shape and interlocking, rather than the actual quartz content (Merriam et al., 1970). Samples with low mica content (Figure 5.3.13) and mica content ~10% (Figure 5.3.14) show a pronounced UCS power function with quartz to feldspar ratio, with levelling of the strength with increasing quartz to feldspar ratio. This further supports the conclusion from Figure 5.3.9 that mica content is an important factor, but also shows that when little mica is present (<10%) the ratio of quartz to feldspar plays a more important role in UCS strength than when ~10% mica is present.

Figure 5.3.13: UCS versus quartz to feldspar ratio categorised according to mica content <10%.
5.3.2.2 Grain Size Sensitivity

The results in Figure 5.3.15 show a similar relationship between grain size and UCS as with Brazilian tensile strength. A negative power relationship between grain size and UCS exists, when both samples with and without grain boundaries are examined together. A negative relationship between grain size and UCS has also been found for granitic rocks from Turkey (Tugrul and Zarif, 1999). The damage initiation and coalescence thresholds show similar relationships with grain size for samples with and without grain boundaries (Figure 5.3.16). The coalescence has a stronger negative relationship with grain size than the initiation for mid-sized grains, as shown by the trend of the ratios between initiation and coalescence threshold stress values (Figure 5.3.16).

An examination of the failure patterns of the samples with and without grain boundaries in Appendix E.4 shows that a large amount of failure initiates with slip in favourably oriented mica grains, which then initiates tensile fractures through feldspar and quartz grains. The relationship between grain size and UCS for samples without grain boundaries is similar to the relationship with Brazilian tensile strength and arises from the improved fracture coalescence and propagation through large grains, as shown by the strong relationship between coalescence and
grain size. The stronger grain size dependence of coalescence and peak strength in physical laboratory testing of granites compared to initiation has been used to suggest that the increased ease of fracture propagation through larger grains is responsible for lower peak strength, more so than increased fracture initiation (Eberhardt et al., 1999b).

In samples with grain boundaries failure also initiates at the grain boundaries and propagates through feldspar, and to a lesser extent, through quartz grains, especially in the samples with smaller grain size (G2 and G3). This is also shown by greater numbers of acoustic emissions in these samples. This initiation at grain boundaries explains the difference in the results between the two datasets. The weak relationship between grain size and UCS for samples with grain boundaries is likely due to the competing influences of improved fracture propagation through larger grains versus decreased initiation locations due to reduced grain area to grain boundary ratio.

![Graph showing UCS versus grain size categorised according to presence of grain boundaries.](image-url)

Figure 5.3.15: UCS versus grain size categorised according to presence of grain boundaries.
5.3.2.2.3 Fabric Type and Intensity Sensitivity

The results in Figure 5.3.17 show decreasing trends with increasing fabric intensity for the samples with schistosity and cleavage, and an increasing trend for samples with mineral preferred orientation, similar to the relationship with PLT index strength for SAG in Section 3.3.6.1. In addition, the samples with cleavage, in general, had higher UCS values than the samples with schistosity. The samples with mineral preferred orientation displayed failure patterns (Appendix E.4) more similar to the samples without fabric than their counterparts with schistosity and cleavage. The mica grains, although aligned, are not continuous across the sample and do not provide a clear path for failure propagation. Instead the micas slip along cleavage planes and induce tensile fractures in the adjacent feldspar grains. The sample with narrow mica spacing is slightly stronger than the sample with narrow spacing likely due to coalescence of tensile fractures into a shear plane, compared with the propagation of tensile macro fractures in the sample with widely spaced micas.

The decrease in UCS with increase in fabric intensity is related to the increase in mica content, as well as the increase in connectivity of aligned mica grains. More localisation of the
failed elements in the domainal samples (as represented by the failure maps of samples with cleavage and schistosity, as well as the lower percentage of failed elements) is likely the cause of the higher UCS values, whereas the localisation decreases with increased fabric intensity. In the domainal samples the fractures have fewer clear failure paths to follow and in most cases must propagate through feldspar and quartz grains.

Despite having lower mica content than the samples with cleavage, the samples with schistosity have lower UCS values. This may be due to the larger size of mica grains, which can accommodate greater strains during slip along the cleavage planes, and thus induce greater numbers of failed elements and lead to failure at lower applied stress.

These results show that increased mica alignment and connectivity, as well as increase in mica content, stemming from different fabric types and intensities will negatively impact the UCS. This has also been shown by a negative relationship between physical laboratory differential biaxial peak strength and linear mica continuity of schists and gneisses (Shea and Kronenberg, 1993). In addition, size of mica grains will negatively impact the UCS. In the case of aligned, but discontinuous micas, very little impact on failure mode is observed; however, the UCS is lower than for samples with similar mica content without alignment.

![Graph showing UCS versus fabric type categorised according to fabric type and intensity aligned at 30° to the axis of loading.](image)

Figure 5.3.17: UCS versus fabric type categorised according to fabric type and intensity aligned at 30° to the axis of loading.
5.3.2.3 Comparison Between Brazilian Tensile Strength and Uniaxial Compressive Strength Results

Two indices based on UCS and Brazilian tensile strength were used to investigate the impact of each of the geological factors on measures of brittleness based on laboratory strength test values. The factors were compared to the ratio between Brazilian tensile strength and UCS (Kahraman, 2002) and the product of UCS and Brazilian tensile strength (Kahraman and Altindag, 2004). In general the relationships between the geological factors and the measures of brittleness are more pronounced when compared to the brittleness index represented by the product of UCS and Brazilian tensile strength (Figures 5.3.18-5.3.22).

Figure 5.3.18: Brazilian tensile strength / UCS ratio versus mica categorised according to quartz to feldspar ratio.
Figure 5.3.19: UCS.Brazilian tensile strength product versus mica categorised according to quartz to feldspar ratio.

Figure 5.3.20: Brazilian tensile strength / UCS ratio versus quartz to feldspar ratio categorised according to mica content.
Figure 5.3.21: UCS.Brazilian tensile strength product versus quartz to feldspar ratio categorised according to mica content

Figure 5.3.22: Brazilian tensile strength /UCS ratio and UCS.Brazilian tensile strength product versus grain size categorised according to presence of grain boundaries

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The mica content shows a mildly negative relationship with the ratio of strength values (Figure 5.3.18) depending on the quartz to feldspar ratio, while there is a pronounced negative relationship with the product of strength values (Figure 5.3.19) regardless of quartz to feldspar ratio. The quartz to feldspar ratio shows a negative relationship with the ratio of strength values (Figure 5.3.20), in particular for rocks with low mica content, and is a more pronounced positive relationship with the product of strength values (Figure 5.3.21). This is comparable to a similar analysis (Figure 5.3.23) of laboratory strength testing data of granitic rock with low mica content (4%) and based on published data from Tuğrul and Zarif (1999). The grain size shows a parabolic relationship with both the indices (Figure 5.3.22).

Figure 5.3.23: UCS.Brazilian tensile strength product versus quartz to feldspar ratio categorised according to mica content (data from Tuğrul and Zarif (1999).
5.3.2.4 Mineral Controls on Fracture Behaviour

An examination of the output data files and the screen captures from the numerical models demonstrates some of the mineral controls on fracture behaviour discussed in Section 4.2.2.2. Some of these were discussed in Section 5.2.5.3 as a verification that the model was indeed behaving as has been described in literature. Additional behaviour was shown during the numerical modelling. An examination of the rate of element failure (analogous to acoustic emissions) classified by mineral type is shown in Figure 5.3.24. A higher percentage of elements designated as quartz have failed compared to any other mineral type. In terms of the three main mineral types, quartz, feldspar and mica have decreasing percentage of elements failed, in that order. This is similar to findings by Wong (1982) describing quartz grains as shattered, feldspar grains as containing many fractures and biotites as being simply kinked in the post-peak region. In focussing on grain boundaries, the quartz grain boundaries are most failed, followed by the feldspar boundaries and the mica boundaries, which have the least percentage of failed elements of all mineral types. This supports the findings in Li (2001) stating that cracks will propagate through quartz boundaries, can propagate through feldspar boundaries and are usually arrested at biotite boundaries and may suggest that quartz and feldspar boundaries are fracture initiators, as in Kranz (1983).

By microscope thin section examination of stressed granite samples Li (2001) determined that quartz grains begin to fracture at 50% of UCS while feldspars begin to fracture at 75% of UCS. This was investigated in modelled results (Figure 5.3.25) and shows an increase in the rate of in quartz element failure at 50% of UCS, continued increase in the rate of feldspar element failure and a sharp increase in the rate of mica element failure at 75% of UCS. Although all mineral types show element failure before the thresholds described by Li (2001), the fact that the rate of element failure increases at these points suggests that visible microfractures are developing at these stresses. The rate of element failure on a mineral basis greatly depends on the mineralogy of the rock (for example Figure 5.3.26). Sample M1 has mineralogy similar to the Lac du Bonnet granite (Diederichs, 1999) and the failure of elements on a mineral basis (Figure 5.3.27) is similar to the findings from Eberhardt et al. (1999a) in which about 50% of microfractures are in feldspar grains at the damage initiation stress. This trend will differ with different mineral ratios, for example sample Q4 has roughly 2/3 of microfractures in feldspar grains at the damage initiation stress, depending on the strength, hardness and stiffness contrasts, as suggested by Eberhardt et al. (1999a).
Figure 5.3.24: Element failure count classified by mineral type of each element, as a percentage of the total number of elements of each mineral type. Sample Q4: 12% mica, 15% quartz, 73% feldspar.

Figure 5.3.25: Element failure fraction for three main mineral types, quartz, feldspar and mica, compared to UCS. Crosses intersect UCS at 50% and 75%. Sample Q4: 12% mica, 15% quartz, 73% feldspar.
Figure 5.3.26: Element failure fraction for three main mineral types, quartz, feldspar and mica, compared to UCS. Sample M1: 6% mica, 19% quartz, 75% feldspar.

Figure 5.3.27: Element failure values for three main mineral types, quartz, feldspar and mica, compared to UCS. Sample M1: 6% mica, 19% quartz, 75% feldspar.
Figure 5.3.28 demonstrates the state of element failure at approximately 50% (left) and 75% (right) of UCS. Quartz and feldspar show the onset of failure at 50% of UCS but microfractures are much more developed at 75% of UCS, especially in the feldspars. Wong (1982) found that high angle cracks occur in mica and feldspar, defined by cleavage, which is demonstrated for mica in the extreme case in Figure 5.3.29. Here shear and tensile failure along ubiquitous joints, the manifestation of cleavage in the model, occur in two mica grains favourably oriented for each failure type. A third mica in the centre of the sample is conspicuously intact as its cleavage is in an unfavourable orientation for failure initiation. A favourably oriented feldspar grain in the lower left of the images in Figure 5.3.30 shows shear failure, although other feldspar grains also show this type of failure in both Figures 5.3.29 and 5.3.30. This shows that failure along cleavage is more important in mica than in feldspar. These are absent in quartz, which tends to fail in tension rather than shear. The feldspar grain boundaries in Figure 5.3.29 failed before the interior failed, as discussed by Li (2001) and Kranz (1983).

Figure 5.3.28: FLAC output of mineral types and failure mode at 50% (left) and 75% (right) of UCS. Micas = teal, feldspars = purple and quartz = pink/red. Failure legend in Figure 5.2.28. Sample M2.
Figure 5.3.29: Element failure (left), mineral type (centre), and shear strain (right) output from post-peak FLAC model. The blue marks in the left image are shear failure along ubiquitous joints while the light green and turquoise marks are tensile failure along ubiquitous joints. Both are in mica grains (right). Failure legend in Figure 5.2.28. Sample G5.

Figure 5.3.30: Element failure (left), mineral type (right) and shear strain (right) output from post-peak FLAC model. The pink marks in the lower left image are shear failure in the direction of cleavage in a feldspar grain (right). Failure legend in Figure 5.2.28. Sample MPO1.
Near peak strength, micas have been observed to kink and contribute to conspicuous cracks in specimens (Tapponier and Brace, 1976) and have been found to initiate fractures in adjacent grains (Li, 2001). This can be seen at 95% of UCS in Figure 5.3.31. Although kinking was not explicitly modelled, the high strains in mica grains and adjacent cracks are analogous to the kinking and cracking described by Tapponier and Brace (1976).

Local tensile stresses were found to develop in Westerly Granite at the interfaces of different mineral grains due to their differences in elastic properties (Tapponier and Brace, 1976). Similar behaviour was found in the modelled rocks, in particular between mica and feldspar grains (Figure 5.3.32). Micas are often zones of higher compressive stress and strain due to their lower shear modulus, inducing tensile stress zones in adjacent feldspars. Under the correct tensile stress conditions, tensile fractures propagate through the feldspars. Elastic properties, and in particular, differences in elastic properties are critical to the promotion of tensile fracture in compression.

Figure 5.3.31: Element failure (left), mineral type (right) and shear strain (right) output from FLAC model at 95% of UCS highlighting higher strain (purple) in mica grains, which initiates tensile fractures in adjacent feldspar grains. Failure legend in Figure 5.2.28. Sample G4.
Figure 5.3.32: Bulk modulus (left) and minor principal stress (right) screen captures with failed elements. Central mica grain (teal, left) with lower shear modulus is under higher compressive stress (purple, right) and is straining more than the adjacent feldspar grains (purple/burgundy, left) inducing tensile stress zones in the feldspar (dark green, right) and promoting tensile fracture (purple circles). Failure legend in Figure 5.2.28. Sample M2.

5.3.3 Classification and Regression Analysis

The Receiver/Response Operating Characteristic (ROC) curve generating process used in Section 4.3 to quantify the geomechanical characterisation scheme F-Factors with respect to spalling sensitivity as identified using TBM performance data was applied to the modelled UCS test results. Each UCS sample was classified as “Spall” or “No Spall” (see Appendix E.4) and each F-Factor was then used to create a ROC curve in order to determine the relative impact of each factor, as shown by the area under the ROC curve, as well as the threshold at which a behaviour change is identified.

5.3.3.1 Mineralogy Sensitivity

The only mica type used in the numerical modelling tests was biotite; therefore, the only relationship with spall sensitivity that is discernable is mica content. The area under the ROC curve in Figure 5.3.33 is 0.82, which demonstrates that mica content has a good accuracy for predicting spalling sensitivity of modelled UCS tests. Rocks with greater than 10% mica fall into the mixed behaviour category, while rocks with greater than 24% mica fall into the spalling
insensitive category and rocks with less than 10% mica fall into the spalling sensitive category. This arises due to the differing impact of mica grains on fracture initiation and propagation. Micas act as both fracture initiators and inhibitors, as discussed in Section 5.2. At mica contents less than 10% micas initiate fractures, but are not in high enough concentration to inhibit propagating fractures, enabling spalling type failure. At content between 10-24% micas act as both fracture initiators and inhibitors, and have a varying impact on spalling failure. At content greater than 24% the mica grains are in such high concentration that the rock failure is governed by mica grain failure, which is preferably shear failure between mica plates.

The area under the ROC curve for quartz to feldspar ratio (Figure 5.3.34) is 0.88, which demonstrates that quartz to feldspar ratio has a good accuracy for predicting spalling sensitivity of modelled UCS tests. Rocks with 0.25-3.45 quartz to feldspar ratio fall into the mixed behaviour category, while rocks with greater than 3.45 quartz to feldspar ratio fall into the spalling sensitive category and rocks with less than 0.82 quartz to feldspar ratio fall into the spalling insensitive category.

![ROC curve for mica content](image)

Figure 5.3.33: ROC curve for mica content
5.3.3.2 Grain Size Sensitivity

Grains with and without grain boundaries were used to investigate the relative impact of grain size on spalling sensitivity of modelled UCS tests. The ROC curve in Figure 5.3.35 has an area of 0.98, which demonstrates that the accuracy of grain size for spalling sensitivity prediction is excellent. The grain size threshold from this dataset is between 4mm and 8mm. A more specific threshold is not possible because tests were not conducted with grain sizes between these two values.
5.3.3.3 Fabric Type and Intensity Sensitivity

The generation of ROC curves requires numerical values, which are not available for fabric type and intensity. To overcome this, each fabric type and intensity was identified with a value between 1 and 3, and ROC curves were generated based on these data. Both ROC curves for fabric type and intensity (Figures 5.3.36 and 5.3.37) have areas below 0.6 and demonstrate that neither this method of classifying fabric is not acceptable for predicting spalling sensitivity of modelled UCS tests.
Figure 5.3.36: ROC curve for fabric type

Figure 5.3.37: ROC curve for fabric intensity
In order to address the lack of values for ROC curve analysis available for fabric type and intensity, a similar approach was taken as in Section 4.3.2.3.2 to develop an estimate for $F_A$. The values for $F_A$ were modified for application to spalling sensitivity rather than chipping and were used to create ROC curves for spalling sensitivity (Figure 5.3.38). The ROC area of 0.85 suggests that the modified $F_A$ factor is well suited to predicting spalling sensitivity of modelled rocks.

5.3.4 Summary of Calibrated F-Factors for Spalling Sensitivity

5.3.4.1 F-Factor Thresholds

The thresholds for each F-Factor in the geomechanical characterisation scheme developed in Section 4.2 by literature review, were modified by the numerical parametric analysis and subsequent ROC analysis. Amphibole, pyroxene, calcite and olivine were not modelled, therefore, the major mineral characterisation is based only on the relative mica, quartz and feldspar contents. Table 5.3.3 demonstrates the revised thresholds for spalling sensitivity within the major mineral category, $F_{MM}$, based on relative quartz content.
Table 5.3.4 demonstrates the revised thresholds for accessory mineral content. The threshold for the accessory minerals group includes the clear threshold for high spalling sensitivity in samples with accessory mineral content less than 10%, mixed spalling sensitivity (characterised as ‘medium’) between 10-24% and low spalling sensitivity above 24% (Table 5.3.4). Only biotite was modelled, and as such no differentiation within the accessory mineral category is possible with this dataset. The discussion regarding the likelihood of plutonic rocks with mica content greater than 10-15% from Section 5.3.2.2.1 suggests that data above this mica content is not applicable and is only presented here for thoroughness. The combined thresholds for the mineralogy factor $F_M$ is shown in Table 5.3.5.

The thresholds for grain size are outlined in Table 5.3.6. The range between 4mm and 8mm grain size has mixed spalling sensitivity and is designated with ‘medium’ sensitivity, and a more precise threshold is not possible with this dataset. The grain size distribution was not tested within this dataset.

The thresholds for spalling sensitivity for fabric are based both on fabric type and spacing (or intensity), as described in Section 4.2. Only cleavage, schistosity and mineral preferred orientation were modelled. The spalling sensitivity designations for fabric type and intensity according to the modified $F_A$ discussed in Section 5.3.3.3 are shown in Table 5.3.7. The designations are recorded as low, medium and high, although within each designation (which follows the fabric intensity) the impact varies according to fabric type.

**Table 5.3.3: Spalling sensitivity designation for major minerals according to relative quartz content ($F_{MM}$)**

<table>
<thead>
<tr>
<th>Spalling Sensitivity Designation</th>
<th>Felsic - Mafic (Quartz to feldspar ratio)</th>
</tr>
</thead>
<tbody>
<tr>
<td>High</td>
<td>&gt;3.45</td>
</tr>
<tr>
<td>Medium</td>
<td>0.82-3.45</td>
</tr>
<tr>
<td>Low</td>
<td>&lt;0.82</td>
</tr>
</tbody>
</table>

**Table 5.3.4: Spalling sensitivity designation for minor minerals according to accessory mineral content and which mineral makes up the greatest proportion ($F_{MA}$)**

<table>
<thead>
<tr>
<th>Spalling Sensitivity Designation</th>
<th>Minor Minerals Content</th>
</tr>
</thead>
<tbody>
<tr>
<td>High</td>
<td>&lt;10%</td>
</tr>
<tr>
<td>Medium</td>
<td>10-24%</td>
</tr>
<tr>
<td>Low</td>
<td>&gt;24%</td>
</tr>
</tbody>
</table>
Table 5.3.5: Spalling sensitivity performance designation for mineralogy factor according to accessory and major mineral content (F_M)

<table>
<thead>
<tr>
<th>Spalling Sensitivity Designation</th>
<th>F_M</th>
<th>&gt;3.45</th>
<th>0.82-3.45</th>
<th>&lt;0.82</th>
</tr>
</thead>
<tbody>
<tr>
<td>High</td>
<td>High</td>
<td>Medium-High</td>
<td>Medium</td>
<td></td>
</tr>
<tr>
<td>Medium</td>
<td>High-Medium</td>
<td>Medium</td>
<td>Medium-Low</td>
<td></td>
</tr>
<tr>
<td>Low</td>
<td>Medium</td>
<td>Medium-Low</td>
<td>Low</td>
<td></td>
</tr>
</tbody>
</table>

Table 5.3.6: Spalling sensitivity designation for grain size (F_GP)

<table>
<thead>
<tr>
<th>Spalling Sensitivity Designation</th>
<th>Grain Size F_GP</th>
</tr>
</thead>
<tbody>
<tr>
<td>High</td>
<td>&gt;8mm</td>
</tr>
<tr>
<td>Medium</td>
<td>4-8mm</td>
</tr>
<tr>
<td>Low</td>
<td>&lt;4mm</td>
</tr>
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</table>

Table 5.3.7: Spalling sensitivity designation for fabric type and intensity (F_A)

<table>
<thead>
<tr>
<th>Spalling Sensitivity Designation</th>
<th>Close spacing</th>
<th>Intermediate</th>
<th>Domainal</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cleavage</td>
<td>Low</td>
<td>Medium</td>
<td>High</td>
</tr>
<tr>
<td>Schistosis</td>
<td>Low</td>
<td>Medium</td>
<td>High</td>
</tr>
<tr>
<td>Mineral Preferred Orientation</td>
<td>Low</td>
<td>Medium</td>
<td>Medium</td>
</tr>
</tbody>
</table>

5.3.4.2 F-Factor Weightings

The best predictors for spalling sensitivity are the grain size, the relative quartz and feldspar contents, and fabric type and intensity followed by the accessory mineral content. The grain size factor F_GP has the highest ROC curve area of 0.98, and therefore the highest weighting. The fabric factor F_A has a ROC area of 0.85, and has the second highest weighting. The quartz content (F_M) relative to feldspar has a ROC curve area of 0.88, followed closely by mica content (F_MA) with a ROC curve area of 0.82. The combined mineralogy factor F_M, therefore, has a combined weighting comparable to fabric.
5.4 Summary of Numerical Calibration of Geomechanical Characterisation Scheme

5.4.1 Numerical Modelling and Constitutive Model

A series of numerical models were created to simulate laboratory strength tests in two dimensions. In order to perform a parametric analysis of the geological factors comprising the Geomechanical Characterisation Scheme using these strength test models a methodology was developed to explicitly simulate rock textures and geological characteristics at the grain size. This methodology consists of separate constitutive models for mica, quartz and feldspar, as well as a method by which each mineral type is simulated in a realistic rock aggregate analogue based on thin sections of rocks from the Gotthard tunnel.

5.4.2 Parametric Analysis of the Geological Characteristics

Parametric analysis of the impact of changing geological characteristics was undertaken using the UCS and Brazilian tensile strength test models. The Brazilian tensile strength test models were used primarily to determine the impact of changing characteristics on the strength values, the behaviour of the tensile fracture through the tested simulated sample and to compare with UCS as a measure of brittleness. The UCS test models were also used for the same purpose as the Brazilian tensile strength test models. In addition, the UCS samples were evaluated for their spalling sensitivity based on the behaviour of the induced fractures during sample yield.

5.4.2.1 Summary of Parametric Analysis Results

The parametric analysis of the impact of different F-Factor combinations on both Brazilian tensile strength and UCS models has provided considerable insight into the failure mode, propagation and ultimate strength values in the following manner:

- The majority of failure in the Brazilian tensile and UCS models is tensile, which has also been observed in physical laboratory testing of granitic rocks (Howarth, 1987; Tapponier and Brace, 1976; Wong, 1982).
- Increased mica content leads to a negative power relationship with both Brazilian tensile strength and UCS because it contains a weakness direction along which slip and strain can occur. Large strains in micas can initiate fractures through adjacent
feldspar and quartz grains, which was also observed in finite element modelling of granitic rock (Li et al., 2003).

- At mica contents less than 10% micas initiate fractures, but are not in high enough concentration to inhibit propagating fractures, enabling spalling type failure. At content between 10-24% micas act as both fracture initiators and inhibitors, and have a varying impact on spalling failure. At content greater than 24% the mica grains are in such high concentration that the rock failure is governed by mica grain failure, which is preferably shear failure between mica plates.
- Increased quartz to feldspar ratio leads to a positive relationship with Brazilian tensile strength and UCS even though feldspar is slightly stronger in shear and tension, suggesting cleavage in feldspar may contribute to decreased UCS. This relationship is more pronounced in samples with mica content less than 10%.
- Simple increase in grain size facilitates tensile fracture coalescence and propagation and reduces both Brazilian tensile strength and UCS, as seen in finite element modelling of granitic rock (Li et al., 2003).
- Grain boundaries can act as fracture arrestors around feldspar grains, as observed by Li (2001).
- Scale dependence of large-grain samples results in Brazilian tensile strength approaching the highest percentage mineral tensile strength with increasing grain size, or negates the effect of increased grain size in UCS.
- Tensile strength has been shown to decrease to a greater extent with increased grain size than compressive strength (Onodera and Asoka-Kumara, 1980), likely due to presence of microfractures, which are not present in the models, and thus do not contribute to strength dependency on grain size.
- Tensile failure propagation in Brazilian tensile testing is easiest through aligned and continuous mica grains, more difficult when they must fracture through feldspar bridges and most difficult when they must fracture almost entirely through feldspar and quartz grains.
- The alignment and continuity of mica grains ease tensile fracture propagation in Brazilian tensile samples aligned parallel to the loading axis.
- Rapid fracture coalescence and localisation of strain decreases Brazilian tensile strength and is negatively related to mica content in Brazilian tensile samples with fabric oriented perpendicular to the loading axis.
• Increased mica content, alignment and connectivity, as well as mica grain size negatively impact the UCS.
• Shear fracture generation is more difficult than tensile fracture propagation in moderately aligned fabric, resulting in slightly higher UCS.
• The comparison of geological factor changes with two brittleness indices suggests that an index based on the product of UCS and Brazilian tensile strength results in a pronounced relationship. This index could be successfully used to estimate the brittleness of the material in question.

5.4.2.2 Summary of Spalling Sensitivity and Brittleness Analysis

The ROC curves for spalling sensitivity during modelled UCS testing provide an indication of the relative impact of mica content, quartz to feldspar ratio, grain size and fabric type and intensity on spalling sensitivity, and the threshold at which spalling sensitivity becomes important. These tests demonstrate that grain size is the leading indicator for spalling sensitivity, followed by quartz to feldspar ratio and mica content. In these tests, fabric type and intensity, which were quantified using an $F_A$ weighted for spalling sensitivity, can be used successfully in combination as predictors for spalling sensitivity during modelled UCS testing.

The comparison with brittleness indices shows that the relationships with the ratio between Brazilian tensile strength and UCS is not very pronounced, as was observed in published real laboratory test data in Chapter 4. The relationships with the product of the Brazilian tensile strength and UCS are well defined, as shown for limited published data. This brittleness index represents the area below the Brazilian-UCS curve, rather than the slope of the curve represented by the ratio between the two strength values. The meaning of each of these measurements will be discussed in detail in Chapter 6.
Chapter 6: Effect of Spall Sensitivity on Rock Cutting and Tunnel Face Stability

6.1 Introduction

6.1.1 Geomechanical Characterisation Scheme for Chipping Performance Prediction

Chipping, as it was determined using back-analysis of TBM data in Chapter 3, is analogous to small spalls, and is sensitive to rock strength and failure behaviour. A weak rock can be as easily excavated as a harder rock that is sensitive to spalling. The distinction between weak rocks that are easily excavated and hard rocks that easily spall and thus are also easily excavated is not well defined. In order to separate these two behaviour types, only rocks with point load index test (PLT) values greater than 2.5 MPa (unless they had low PLT values due to discing) were used in the chipping performance analysis in Section 3.3. For this reason, good chipping performance can be considered as a function of both rock strength and spalling sensitivity, as in Equation 6.1.1.

\[ Chipping\ Performance = f(Spalling\ Sensitivity, \ Strength) \]  

This equation is based on the premise illustrated in Figure 6.1.1, in that both are a measure of TBM performance based on normalised strength. Equation 6.1.1 addresses the question of chipping performance only, while Figure 6.1.1 addresses the interrelated aspects of chipping performance and face stability as they relate to the net advance rate. An increase in the chipping performance of a rock will move it to the right in Figure 6.1.1, as will an increase in in-situ stress. An increase in the in-situ stress can improve the chipping performance of certain rock types that have a higher spalling sensitivity by taking advantage of spalling-type failure.

Numerical modelling of the cutting process was undertaken to investigate the impact of different geomechanical characteristics in simulated rocks on the chipping process. These tests were related to the spalling sensitivity testing undertaken in Chapter 5 to obtain a relationship between spalling sensitivity and chipping performance, within the framework of Equation 6.1.1.
6.1.2 Geomechanical Characterisation for Boundary Stability

The interrelation of rock strength and in-situ stress condition emphasises the importance of Figure 6.1.1 in anticipating TBM performance from geological and in-situ stress condition information. Face instability, as defined in Chapter 3, was investigated using numerical modelling of the cutting process with variations on in-situ stress conditions. The results were used to investigate the relationships between rock strength, spalling sensitivity, in-situ stress condition and chipping performance.

6.1.3 Geomechanical Characterisation

The thresholds and weightings determined using the ROC analysis of geological and TBM data were used to adjust the classification scheme from Section 4.2 for chipping performance based on geomechanical data. The framework from the Geomechanical Characterisation Scheme was used to predict chipping performance and face instability by using the thresholds determined with the ROC analysis of the F-factors in Section 4.3. Weightings were slightly modified to reflect the relative importance of each F-factor determined by the ROC.
curve analysis. The resulting characterisation scheme was used to modify the side maps from Appendix B.4, and is presented in Appendix F.3.

Indicators of brittleness and failure behaviour were discussed in Section 4.1. Some of the indicators were found to relate to the type of data available through the numerical modelling undertaken in Chapter 5, and were found to be applicable to the conditions present during TBM excavation. These tools are based on laboratory test values that are typically easily obtainable. Their availability is their biggest advantage and makes them valuable for the translation of the information available from numerical modelling of laboratory tests and the cutting process. These numerical results could then be used as indicators and analogues of real rock behaviour and provide the relationships needed to make chipping performance and face stability predictions from geomechanical characteristics and information regarding the anticipated in-situ stress condition.

6.1.4 Spalling Sensitivity Factor

The spalling limit, represented as a ratio of $\sigma_3/\sigma_1$ from biaxial or triaxial strength testing, is dependent on the degree of heterogeneity, and external factors, such as stress rotation and rock damage, while the systematic damage initiation threshold is dependent on geomechanical properties of the rock (Diederichs et al., 2004). In particular an increase in heterogeneity, high damage levels and unfavourable stress conditions will result in a shallower spalling limit curve, thus allowing for greater potential for spalling failure (Figure 6.1.2). In addition, geomechanical properties that lower the systematic damage initiation threshold will also allow for greater potential for spalling failure. Investigating the geomechanical dependence of the spalling limit and the systematic damage initiation threshold in a testable configuration was a major goal of this thesis, and the spalling sensitivity factor $F_{SS}$ introduced in Chapter 4 was tested for its ability to accomplish this goal.
6.1.4.1 Defining the Spalling Sensitivity Factor for Numerical Modelling Tests

The modelled tests introduced in Section 5.2.5.3.3 were compared to laboratory test data (Figures 6.1.3 and 6.1.4) in terms of ratio of initiation stress to peak stress and initiation stress to coalescence stress. The methods by which initiation and coalescence were identified were different for the laboratory and modelled tests, as discussed in Section 5.2.5.3.3. In addition, for each test dataset, two different methods were used to identify the initiation, resulting in a lower and a higher estimate rather than a single value. These are reflected in the four sets of ratios presented in Figures 6.1.3 and 6.1.4.

Some issues with the numerical modelling were discussed in Section 5.4, in particular with respect to the meaning of the peak value in a numerical model compared to the meaning of the peak value in laboratory tests. These issues were discussed but not resolved in this research, and as such Figure 6.1.3 contains ratios with respect to peak while Figure 6.1.4 contains ratios with respect to coalescence. It has been suggested (Diederichs, 1999) that two-dimensional models can really only simulate UCS testing up to coalescence, making the ratio between initiation and coalescence more useful in this analysis. The hoop stresses discussed in Section 4.1.3 impact laboratory peak stress values, and it has been suggested that coalescence is the onset
of sample yield (Diederichs et al., 2004), making a ratio between initiation and coalescence a more representative indicator of failure behaviour during UCS testing.

The modelled values in Figure 6.1.4 show a more defined increasing trend with the laboratory values. Figures 6.1.5 and 6.1.6 show comparisons between laboratory and modelled ratios, compared to a reference line with slope of 1. The comparison between the low initiation/coalescence ratios for both real and modelled values provides the trend slope most closely approaching 1 (Figure 6.1.5), with an $R^2$ value of 0.34 indicates poor correlation. This, though, is the highest correlation value for the initiation/coalescence ratio comparisons. Higher correlation values exist for initiation/peak ratio comparisons, however, these correlations are for slopes that are much lower than 1 (Figure 6.1.6). This shows that when comparing laboratory and model UCS results, the ratios of lower bound initiation values to coalescence values result in the best (though not good) correlation.

Figure 6.1.3: Laboratory and modelled initiation/peak ratios of UCS tests in order of increasing “laboratory – lower initiation/peak” ratio.
Figure 6.1.4: Laboratory and modelled initiation/coalescence ratios of UCS tests in order of increasing “laboratory – lower initiation/coalescence” ratio.

Figure 6.1.5: Comparison of laboratory to modelled initiation/peak ratios of UCS tests combined in terms of the higher and lower initiation values, with real values listed first (i.e. high real – high modelled). 1:1 ratio line shown for comparison.
6.1.4.2 Application of the Strength Reduction Factor to Rocks with and without Fabric

The preliminary methodology developed by Diederichs et al. (2004) to quantify geological characteristics for excavation boundary strength estimation was applied to the samples from Section 5.3.3.1, which were tested for UCS (see Appendix B.3), then analysed by thin section and characterised for F-Factors. The ratios of initiation/coalescence and coalescence were compared to the Field Strength Ratio (FSR) (Table 6.1.1 and Figures 6.1.7 and 6.1.8).

The data points in Figure 6.1.8 were fit with trendlines forced through the origin to examine the relationships between the FSR and laboratory data ratios. The forced trendline for the lower initiation/coalescence has a slope of 1.03 with an $R^2$ of 0.19, while the forced trendline for the higher initiation/coalescence has a slope of 1.17 with an $R^2$ of 0.13. A visual examination shows that the cloud of data points for the lower ratios are scattered around the forced trendline with a slope of nearly 1. A similar investigation of the initiation/peak ratios from Table 6.1.1 provided similar results using forced trendlines, with slopes deviating more from 1 than the
initiation/coalescence ratios. This shows that the initiation/coalescence ratio is a better ratio to use when comparing laboratory strength test data to both modelled data (as shown in Section 6.1.3.1) and estimates based on geomechanical properties (for example FSR).

Table 6.1.1: Estimation of lower bound, upper bound and field strength ratio (FSR) proposed by Diederichs et al (2004).

<table>
<thead>
<tr>
<th>Sample</th>
<th>UCS</th>
<th>$\sigma_{ci}$</th>
<th>$\sigma_{cs}$</th>
<th>$\sigma_{ci}/UCS$</th>
<th>$\sigma_{cs}/UCS$</th>
<th>F1</th>
<th>F2</th>
<th>F3</th>
<th>F4</th>
<th>FSR</th>
</tr>
</thead>
<tbody>
<tr>
<td>a065</td>
<td>149</td>
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<td>136</td>
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<td>0.8</td>
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<tr>
<td>a169</td>
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<td>57</td>
<td>139</td>
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<td>0.93</td>
<td>0.8</td>
<td>0.8</td>
<td>0.85</td>
<td>0.85</td>
<td>0.46</td>
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<tr>
<td>b011</td>
<td>135</td>
<td>38</td>
<td>105</td>
<td>0.28</td>
<td>0.78</td>
<td>0.8</td>
<td>0.75</td>
<td>0.85</td>
<td>0.85</td>
<td>0.43</td>
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<tr>
<td>b023</td>
<td>183</td>
<td>56</td>
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<td>0.31</td>
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<td>0.8</td>
<td>0.8</td>
<td>0.85</td>
<td>0.75</td>
<td>0.41</td>
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<td>b039</td>
<td>109</td>
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<td>103</td>
<td>0.44</td>
<td>0.94</td>
<td>0.8</td>
<td>0.75</td>
<td>0.85</td>
<td>0.9</td>
<td>0.46</td>
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<tr>
<td>b088</td>
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<td>0.75</td>
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<td>b124</td>
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<td>114</td>
<td>0.45</td>
<td>0.88</td>
<td>0.8</td>
<td>0.75</td>
<td>0.85</td>
<td>0.9</td>
<td>0.46</td>
</tr>
</tbody>
</table>

Figure 6.1.7: FSR values and real initiation/coalescence ratios of UCS tests in order of increasing “real – lower initiation/ coalescence” ratio.
Two caveats for the FSR methodology are that (1) it is not applicable to highly foliated or schistose rock and that (2) the sample must be undamaged. All of the samples used in this analysis are deemed to be stress damaged, likely due to stress release during sampling at high overburden stresses. This observation is based on the p-wave velocities and the ratio between the tangential Young’s Modulus and secant Young’s Modulus at 75% UCS, which is on average 50% for the dataset (contained in Appendix B.3). These are far below the threshold initially suggested by Diederichs et al. (2004) of 90%. In addition, the only samples without strong foliation are b023 and b088. A comparison of the lower bound ratio to the FSR shows agreement within 15% for only a169, b039 and b124. This raises an issue for testing of damaged core to obtain critical failure threshold values versus estimating the field strength reduction factor based on an examination of the geological characteristics. The FSR methodology (or any methodology based on the characterisation of the geology) would be immune to damage related problems; however, the peak strength values obtained from a UCS test of damaged core is generally reduced by damage. The application of equation 6.1.2 to damaged samples would lead to an underestimate of the lower bound excavation boundary strength (UCS$_{\text{in situ}}$).

\[
\text{UCS}_{\text{in situ}} = \text{FSR} \times \text{UCS}_{\text{lab}} \tag{6.1.2}
\]

Figure 6.1.8: Comparison of FSR to laboratory initiation/coalescence ratios of UCS tests with trendlines fit through the origin to compare to a line with slope=1.
The systematic damage initiation threshold was found to be less sensitive than the peak strength to initial sample damage and sample size in PFC modelled rocks (Diederichs et al., 2004), and can be considered as an intrinsic material property. The high sensitivity of peak strength to initial sample damage arises from the increased potential for interaction due to the increase in number of cracks present. This should also hold true for coalescence. A ratio between the initiation and coalescence is unlikely to be less sensitive to initial sample damage than a ratio of initiation to peak strength, and would not provide a better estimate of in-situ excavation boundary strength.

The FSR itself had a correlation of 0.19 for the ratios between initiation and coalescence for this dataset (Figure 6.1.9 “FSR”) with a linear fit forced through the origin. In an attempt to improve the FSR estimation, a fifth factor (F5) was used to characterise the fabric (Figure 6.1.9 “FSR with Fabric”). The addition of this factor provided an improvement to the estimate to an $R^2$ value of 0.28 for a linear fit forced through the origin, although it is still too low to establish a meaningful correlation. The correlations appear to be significant, however, an observation of the data points with respect to the linear fit forced through the origin shows considerable scatter and suggests that the comparisons are not adequate.

The preliminary work presented in Diederichs et al. (2004) is similar in principle to the Geomechanical Characterisation Scheme developed in Chapter 4. In order to explore the applicability of the characterisation scheme to predict the strength ratios presented above, the rock samples were characterised according to the geomechanical characterisation scheme and the resulting $F_{SS}$ was multiplied by $F_1$ from the Diederichs et al. (2004) FSR system (in this case 0.8). The resulting predicted values are shown in Figure 6.1.9 as “Trial Ratio”. The correlation of the estimated versus calculated ratio values is not significant ($R^2 = -2.3$) for the Geomechanical Characterisation system as it was presented in Chapter 4.
6.1.4.3 Parametric Calibration of the Spalling Sensitivity Factor $F_{SS}$

Although the Geomechanical Characterisation Scheme as presented in Chapter 4 did not provide valid estimates for initiation/coalescence ratios with relevant correlations to the laboratory data ratios from UCS testing, further investigation of the geomechanical characterisation F-Factors was undertaken in Chapter 5 by numerical modelling. The results from the spalling sensitivity analysis, in particular the thresholds and weightings of the factors, were used to adjust the F-Factors such that they would better estimate initiation/coalescence values. The same dataset of numerical model UCS tests used in Chapter 5 was used in this analysis. The samples were characterised according to the Geomechanical Characterisation Scheme and the $F_{SS}$ value was compared to the initiation/coalescence ratios averaged over the three tests for each sample type (Table 6.1.2). The thresholds and weightings determined in Chapter 5 were used as a baseline and the values were then adjusted such that the resulting $F_{SS}$ values fit the entire dataset (Figures 6.1.10 and 6.1.11). The resulting $F_{SS}$ values were found to have a slope of 1 and an $R^2$ value of 0.75 when compared to initiation/coalescence ratio and a slope of 1.2 and an $R^2$ of 0.62 when compared to initiation/peak. The comparison to the ratio of initiation/peak suggests that the estimates are also compatible with the ratio between initiation/coalescence.
<table>
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<tr>
<th>Sample</th>
<th>Average Measured Ratio</th>
<th>$F_{SS}$</th>
<th>$F_I$</th>
<th>$F_G$</th>
<th>$F_{MM}$</th>
<th>$F_{MA}$</th>
<th>$F_A$</th>
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<td>0.66 0.44 0.56 0.67</td>
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<td>0.95</td>
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<td>1.05</td>
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<td>0.76</td>
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<td>0.95</td>
<td>0.9</td>
<td>1.1</td>
</tr>
</tbody>
</table>
Figure 6.1.10: Comparison of calculated initiation/coalescence ratio and estimated $F_{SS}$ value for parametric modelled UCS test results.

Figure 6.1.11: Comparison of estimated $F_{SS}$ to initiation/coalescence and initiation/peak ratios of modelled UCS tests with trendlines fit through the origin to compare to a line with slope=1.
In order to verify that the geomechanical characterisation scheme F-Factors were adequately applicable to data other than the training dataset, the scheme was used to characterise the set of values from the UCS tests from Section 6.1.3.2 and Table 6.1.1. Figures 6.1.12 and 6.1.13 show the results of the comparison both to the laboratory UCS test data and the numerical model UCS test data. In both cases the estimate is imperfect, but the correlations shown in Figure 6.1.14 (“Parametric Adjusted” ratios) show that this is a more significant estimate than previous attempts (i.e. Figure 6.1.9), with R² values of –0.23 and –0.18 for the laboratory and modelled data, respectively. These correlations are lower than those for Figure 6.1.9, however, an examination of the data points shows that they are much less scattered around the linear fit forced through the origin, and are in effect an improvement. The F-Factors were also adjusted specifically for this dataset (“Adjusted” ratios), with improved correlations of 0.58 and –0.5, respectively. This small dataset, however, without considerable variation in the geomechanical characteristics, made it impossible to apply these adjusted values to the more comprehensive dataset of initiation/coalescence ratios from Table 6.1.2, and the F-Factors calibrated with the parametric dataset were retained.

Figure 6.1.12: Estimated FSS values adjusted according to the parametric analysis of modelled UCS tests and initiation/coalescence ratios of laboratory UCS tests in order of increasing “Parametric Adjusted Ratio” ratio.
Figure 6.1.13 Estimated F_{SS} values adjusted according to the parametric analysis of modelled UCS tests and initiation/coalescence ratios of modelled UCS tests in order of increasing “Parametric Adjusted Ratio”.

Figure 6.1.14: Comparison of estimated adjusted initiation/coalescence ratios to laboratory and modelled initiation/coalescence ratios of UCS tests with trendlines fit through the origin to compare to a line with slope=1.
Gong and Zhao (2007) found that increased brittleness in fresh granites leads to easier TBM excavation. The ratio used by Gong and Zhao (2007) to represent brittleness has been suggested as an indicator of brittleness by Hucka and Das (1974) and simplified by Kahraman (2002). This ratio between UCS and Brazilian tensile strength (Equation 6.1.2) was modified to Equation 6.1.3, calculated for the modelled UCS test dataset (Table 6.1.2) and compared to the parametric adjusted FSS, as shown in Figure 6.1.15 with a correlation of –0.4 with a linear trendline forced through the origin (Figure 6.1.17). The inverse of equation 6.1.2 (Equation 6.1.4) was modified (Equation 6.1.5) and compared to the FSS (Figure 6.1.16) with a correlation of –0.05 with a linear trendline forced through the origin (Figure 6.1.17). Neither of these is a considerable correlation but, again, they are better than the correlations from Section 6.1.3.3.

\[
B_1 = \frac{UCS}{\sigma_i} \quad 6.1.3
\]

\[
B'_1 = \frac{1}{20} \left( \frac{UCS}{\sigma_i} \right) \quad 6.1.4
\]

\[
B_2 = \frac{\sigma_i}{UCS} \quad 6.1.5
\]

\[
B'_2 = 5 \left( \frac{\sigma_i}{UCS} \right) \quad 6.1.6
\]
Figure 6.1.15: Comparison of measured brittleness ratio $B_1$ and estimated $F_{SS}$ values

Figure 6.1.16: Comparison of measured brittleness ratio $B_2$ and estimated $F_{SS}$ values
Figure 6.1.17: Comparison of estimated $\text{F}_{\text{SS}}$ to brittleness indices of modelled UCS tests with trendlines fit through the origin to compare to a line with slope=1.

A different brittleness definition for laboratory strength testing was suggested by Kahraman and Altindag (2004), by Equation 6.1.7, which measures the area under the Brazilian tensile strength versus UCS graph. This equation was modified in Equation 6.1.8 for comparison to the $\text{F}_{\text{SS}}$ in Table 6.1.2. The comparison is shown in Figure 6.1.18, in which the mineralogy and fabric aspects of the comparison correlate with an $R^2$ factor of 0.44 (Figure 6.1.17).

$$B_3 = \frac{\sigma_i \cdot UCS}{2}$$  \hspace{1cm} 6.1.7

$$B'_3 = 0.3 + \frac{\sigma_i \cdot UCS}{10000}$$  \hspace{1cm} 6.1.8
The $F_{SS}$ are much better matches to the modified product of UCS and Brazilian tensile strength, as calculated by Equation 6.1.7, than it is to the ratio of UCS to Brazilian tensile strength for the modelled data. The correlation is nearly as good as the correlation to the ratio of initiation/coalescence as shown in Figure 6.1.9, suggesting that $F_{SS}$ is also a good predictor of rock brittleness as described by Kahranam and Altindag (2004).

Hucka and Das (1974) state that several indicators of brittleness have been suggested by various authors, including: little elongation or change in sample area, fracture failure and production of fines, higher resilience and ratio between compressive and tensile strength, higher internal friction angle and the formation of cracks under indentation. In this framework, the ratio of systematic damage initiation stress to fracture coalescence stress and the product of UCS and tensile strength are both indicators of brittleness, albeit in slightly different ways. The stress at which systematic damage initiation begins is also the point at which the rate of change of the ratio between lateral and axial strain increases with increased stress. This stress has been suggested as the lower bound excavation boundary strength (Diederichs et al., 2004). The ratio between damage initiation and fracture coalescence stress is representative of brittleness, in that with a lower damage initiation point with respect to fracture coalescence stress (the ratio decreases), tensile damage initiation plays a greater, earlier role in terms of stress. The product of tensile
strength and UCS is representative of brittleness because it demonstrates the balance between the ease with which a sample will rupture under (indirect) tensile stress conditions with the difficulty under compressive stress conditions.

The F_{SS} and the product of UCS and Brazilian tensile strength are representative of brittleness since they generally demonstrate the rapidity (in terms of increasing stress) with which tensile damage is initiated and progresses to coalescence, localisation and rupture in tensile or UCS tests. This is expressed as a lower systematic damage initiation stress to peak stress ratio, a lower UCS to tensile strength ratio and lower UCS and tensile strength product. In Figures 6.1.19 to 6.1.24 the ratios associated with different modelled geomechanical characteristics indicate the following observations:

- Increased fracture tendency is indicated by a decrease in initiation/coalescence ratio, an increase in brittleness factor B_1', an decrease in brittleness factors B_2' and B_3'.
- Brittleness increases with increased unaligned mica content due to the higher proportion of compliant micas deforming and initiating tensile fractures through adjacent feldspar and quartz grains (Figure 6.1.19).
- Brittleness decreases with increased quartz to feldspar ratio due to the lower heterogeneity in strength and stiffness. This is more pronounced in samples with very low mica content (Figures 6.1.20 versus 6.1.21).
- Brittleness generally decreases with increased grain size when grain boundaries are not considered in the numerical model (Figure 6.1.22).
- Brittleness does not greatly vary with grain size when grain boundaries are considered in the numerical model (6.1.23).
- Brittleness generally increases with increased fabric intensity, although the results are ambiguous for rocks with mineral preferred orientation (Figure 6.1.24).
Figure 6.1.19: Comparison of indicators for spalling sensitivity and brittleness to mica content.

Figure 6.1.20: Comparison of indicators for spalling sensitivity and brittleness to quartz to feldspar ratio for samples with very low mica content.
Figure 6.1.21: Comparison of indicators for spalling sensitivity and brittleness to quartz to feldspar ratio for samples with mica content 10-12%.

Figure 6.1.22: Comparison of indicators for spalling sensitivity and brittleness to grain size for samples without grain boundaries.
Figure 6.1.23: Comparison of indicators for spalling sensitivity and brittleness to grain size for samples with grain boundaries.

Figure 6.1.24: Comparison of indicators for spalling sensitivity and brittleness to fabric type and intensity.
6.1.4.4 Spalling Sensitivity Prediction

The $F_{SS}$ described in Section 4.4.3 were used to create a ROC curve to investigate their potential for predicting spalling sensitivity of modelled UCS test samples. Figure 6.1.25 contains the ROC curve, with an area of 0.78, which shows that the $F_{SS}$ is a good predictor of spalling sensitivity. The dot histogram in Figure 6.1.26 provides an interesting view on the relationship between the spalling sensitivity $F_{SS}$ and the designation of spalling versus non-spalling given to the modelled test samples. A threshold value for predicting spalling sensitivity can be picked out from these figures at an $F_{SS}$ value of 0.57. These figures also highlight the tendency that spalling sensitivity, as used to describe modelled UCS tests in Chapter 5, increases with decreasing $F_{SS}$ value.

![ROC curve](image)

Figure 6.1.25: ROC curve for ability of $F_{SS}$ to predict spalling behaviour of the modelled UCS tests making up the parametric analysis dataset.
Figure 6.1.26: Dot histogram of spalling sensitivity designation and $F_{SS}$ values for the modelled UCS tests making up the parametric analysis dataset.

6.1.4.5 Brittleness Indices and F-Factors

The weighted F-Factors were modified to achieve better fits with respect to the initiation and coalescence ratios, and published brittleness indices. They were not correlated to TBM performance, although, they were shown to relate to face instability. The three brittleness indices that were compared to weighted F-Factors showed the best correlation with $B_3'$. The brittleness indices investigated all use combinations of Brazilian tensile strength and UCS where $B_1'$ and $B_2'$ are essentially reciprocals of each other, measuring the slope of the line in a BTS versus UCS graph (Figure 6.1.27 left). $B_3'$ essentially represents the area under the BTS versus UCS curve (Figure 6.1.27 right). This difference prompts the question: what is the fundamental difference and how does that impact prediction of brittleness? Both formulations can have non-unique solutions and while the ratio formulation is a normalization of one strength by the other, the product formulation equally weighs the contribution of each strength. The better data fit arising from the $B_3'$ formulation likely arises from this feature, where BTS and UCS both describe two different aspects of rock yielding behaviour and when put together are better at describing the brittleness.
Both hypothetical behaviours can be zoned according to brittleness behaviour being described (Figure 6.1.27), although the nature of the formulations makes it impossible for behaviour zones to correspond to both brittleness indices. Both formulations are valid, and further testing should be undertaken to determine what the behaviour zones are and identify which formulation best describes brittleness.

6.1.5 **Numerical Modelling of the Chipping Process**

In order to relate the weighted F-Factors for chipping performance and to correlate to the results from Chapter 4 a two-dimensional numerical model was created to simulate the chipping process. Numerical modeling of rock cutting has been undertaken using finite element codes (Liu et al., 2002a; Liu et al., 2002b; Sulem and Cerrolaza, 2002) with various geometries, but in general models consist of a rock block and one or two cutters loaded with a fixed velocity (displacement per step). The goals of the investigations ranged from geometrical studies (Liu et al., 2002b), rock properties (heterogeneity (Liu et al., 2002b), grain size (Sulem and Cerrolaza, 2002)) and fracture behaviour during cutting (Liu et al., 2002a). None of the conclusions from published work were capable of fulfilling the requirements for this research. A two-cutter model was created in FLAC with the ability to simulate rock analogues developed in Chapter 5. Model development and sensitivity analysis are presented in Appendix F.1, but the model characteristics are summarized below.
6.1.5.1 Two-Cutter Numerical Model

The two-cutter numerical model is shown in Figure 6.1.28 and has 0.5mm elements in the rock block. Each cutter is represented by half a cutter and the rock block is 90mm wide by 120mm deep to make computation efficient. Four cutter passes are simulated by incrementally (1mm) moving the cutter into the rock (Figure 6.1.29), then applying a velocity gradient that results in stress at the cutter tip that is comparable to the stress at real 3-D TBM cutter tips. The texture-generating algorithm described in Appendix E.3 was used to simulate rock, while steel properties were used for the cutters (Figure 6.1.30). After four cutter passes (for a total of 4mm penetration) the failed elements (Figure 6.1.31 left) and accumulated plastic strain (Figure 6.2.31 right) are used to determine the chip fracture length, depth and area below each cutter. The area is the length multiplied by the depth, and the total length, total depth and total area are the sum of length, depth and area below each cutter. The chip area was used as the main indicator of chipping performance for analysis of numerical model results. Samples with initial stress condition were first run with elastic properties to determine the in-situ stress at equilibrium after model relaxation.

Figure 6.1.28: Schematic of small-scale two-cutter model showing dimensions and boundary conditions (grid not to scale) without in-situ stress (left) and with in-situ stress (right).
Figure 6.1.29: Schematic of cutter advance in 1mm increments to simulate 1mm deep kerf. The rock grid was cut during grid generation to allow the subsequent progression of the cutters (left to right). The joints subsequently used along cutter boundary (with joint properties) were temporarily attached, then given joint properties at the appropriate time. Other joints not used along cutter boundary (but necessary in grid generation) were attached throughout the simulation.

Figure 6.1.30: FLAC small-scale model outputs showing fabric, in this case schistosity type 1, aligned at 60° to the tunnel face boundary. Red = quartz, yellow = feldspar, teal = mica.
The two-cutter model was used to examine the impact of in-situ stress and geomechanical characteristics on face stability. Different stress conditions were investigated in combinations with different rock textures to examine the impact on fracture behaviour, face stability, penetration and required thrust. These included: isotropic stresses and biaxial stresses as well as different stress magnitudes.
6.2 Geomechanical Characterisation for Tunnel Boundary Stability

6.2.1 Spalling Sensitivity at the Tunnel Boundary

The terms face instability and spalling sensitivity were introduced in Chapters 3 and 4, respectively. Face instability is the failure of parts of the tunnel face arising from the generation of new fractures in the tunnel face leading to the release of rock blocks ahead of the TBM due to the combination of in-situ stress relaxation at the face and the geomechanical characteristics of the rock. This is not instability arising from the release of blocks defined by pre-existing fractures and joints. Spalling sensitivity describes the combination of geomechanical characteristics that lead to rock that fails through spalling and extensile fracture propagation. In general, tunnel boundary stability would describe the stability of the entire tunnel boundary, including the tunnel walls, but other than for demonstration purposes, tunnel wall stability is not the focus of this research.

The impact of face instability on TBM performance was demonstrated for the Southern Aar granite in Chapter 3. Figure 6.2.1 shows the net advance rate for the Southern Aar granite over a distance of 400m. As discussed in Chapter 3, the sections with low NAR in this rock mass arise from face instability (low NAR section at left) and from tough excavation conditions (low NAR section at right). Face instability also occurred during the excavation of the Altkristallin, resulting in decreased net advance rate (Figure 6.2.2) arising from increasing occurrence of and depth of face instability. The very low advance rate values at TM 111300 arose from tough excavating conditions associated with a change in geomechanical characteristics.
Figure 6.2.1: Net advance rate data for Southern Aar granite

Figure 6.2.2: Net advance rate data for Altkristallin
The net advance rate and drillability index were used in Chapter 3 to identify locations in the Southern Aar granite where face instability occurred. The same methodology was used to identify the locations with face instability in the Altkristallin. The progression from tough excavation, through preconditioning to face instability (Figures 6.2.3 and 6.2.4) schematically shows the impact of progressively increasing magnitude of face instability on TBM performance.

In locations with face conditions exhibiting behaviour ranging from preconditioning to face instability, the penetration rate was found to increase with respect to applied thrust. As discussed in Chapter 3, preconditioning of the tunnel face can aid excavation by creating microfractures in the rock ahead of the TBM, which could then be used by the cutters to aid in chipping.

Figure 6.2.3: Schematic showing relationship between interaction of face stress and strength in the rock and the TBM performance.
6.2.2 Numerical Investigation of Impact of Fabric Orientation and In-Situ Stress on Tunnel Face Preconditioning

6.2.2.1 In-Situ Stress Conditions and Stress Relief

The effects of stress condition and fabric alignment on tunnel face preconditioning were examined by in-situ stress condition (Table 6.2.1) in the two-cutter model (results are found in Appendix F.2) for samples without fabric (quartz variation 4) and samples with fabric parallel and 45° to the tunnel face (domainal schistosity). The analysis was undertaken by running the two cutter models without the cutters until the system equilibrated and examining the resulting element failure and plastic strain. The stress paths for each in-situ stress condition were obtained by elastic modelling and are compared to the damage initiation point of UCS tests and theoretical damage initiation thresholds for quartz variation 4 and domainal schistosity (Figure 6.2.5). All stress paths show that face perpendicular stress drops to 0, as expected, since stresses perpendicular to excavation boundaries are always 0 at the boundary. The stress paths also show a decrease in face parallel stresses due to the rock relaxation and strain perpendicular to the tunnel face causing reciprocated strain parallel to the tunnel face related according to Poisson’s ratio.
### Table 6.2.1: Summary of in-situ stress conditions

<table>
<thead>
<tr>
<th>Stress state</th>
<th>In-Situ Stress (Input)</th>
<th>Boundary Stress (Relaxed)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>In-plane stress</td>
<td>Out-of-plane stress</td>
</tr>
<tr>
<td></td>
<td>perpendicular to face</td>
<td>parallel to face</td>
</tr>
<tr>
<td></td>
<td>(MPa)</td>
<td>(MPa)</td>
</tr>
<tr>
<td></td>
<td>In-plane stress</td>
<td>Out-of-plane stress</td>
</tr>
<tr>
<td></td>
<td>perpendicular to face</td>
<td>parallel to face</td>
</tr>
<tr>
<td></td>
<td>(MPa)</td>
<td>(MPa)</td>
</tr>
<tr>
<td>Isotropic very low</td>
<td>10</td>
<td>10</td>
</tr>
<tr>
<td>Isotropic low</td>
<td>15</td>
<td>15</td>
</tr>
<tr>
<td>Isotropic medium</td>
<td>35</td>
<td>35</td>
</tr>
<tr>
<td>Isotropic high</td>
<td>55</td>
<td>55</td>
</tr>
<tr>
<td>Biaxial low</td>
<td>20</td>
<td>12</td>
</tr>
<tr>
<td>Biaxial medium</td>
<td>30</td>
<td>20</td>
</tr>
<tr>
<td>Biaxial high</td>
<td>50</td>
<td>30</td>
</tr>
<tr>
<td>Biaxial low ratio</td>
<td>50</td>
<td>40</td>
</tr>
<tr>
<td>Biaxial medium ratio</td>
<td>50</td>
<td>30</td>
</tr>
<tr>
<td>Biaxial high ratio</td>
<td>50</td>
<td>20</td>
</tr>
<tr>
<td>Biaxial Parallel high</td>
<td>30</td>
<td>50</td>
</tr>
<tr>
<td>Biaxial Parallel medium</td>
<td>15</td>
<td>50</td>
</tr>
<tr>
<td>Biaxial Parallel low</td>
<td>5</td>
<td>50</td>
</tr>
<tr>
<td>No stress change Isotropic very low</td>
<td>0</td>
<td>10</td>
</tr>
<tr>
<td>No stress change Isotropic low</td>
<td>0</td>
<td>20</td>
</tr>
<tr>
<td>No stress change Isotropic medium</td>
<td>0</td>
<td>35</td>
</tr>
<tr>
<td>No stress change Isotropic high</td>
<td>0</td>
<td>55</td>
</tr>
<tr>
<td>No stress change Biaxial very low</td>
<td>0</td>
<td>10</td>
</tr>
<tr>
<td>No stress change Biaxial low</td>
<td>0</td>
<td>20</td>
</tr>
<tr>
<td>No stress change Biaxial medium</td>
<td>0</td>
<td>35</td>
</tr>
<tr>
<td>No stress change Biaxial high</td>
<td>0</td>
<td>55</td>
</tr>
</tbody>
</table>
The samples with the major principal stress parallel to the tunnel face represent the in-situ stress conditions encountered during excavation, simulated on rock that has not been preconditioned with stress paths for that are parallel during relaxation (Figure 6.2.6a). The samples with the major principal stress perpendicular to the tunnel face represent the preconditioning that occurs during relaxation of the face perpendicular stress, and stress paths are all different, arising from a 90° rotation of the stresses after relaxation (Figure 6.2.6b). Both sets of stress conditions do not intersect the damage initiation threshold for either of the samples tested (Figures 6.2.5 and 6.2.6c). Models explicitly simulating different pre-excavation stress paths (Figure 6.2.7) were not performed in this study.

Figure 6.2.5: Stress paths for the modeled stresses where pre-tunnelling stress is the input stress, and post-tunnelling stress is the equilibrium stress. Samples without face-perpendicular input stress (‘no damage’) are shown for comparison.
The face perpendicular stress (Figure 6.2.8a) and face parallel stress (Figure 6.2.8b) each have different impacts on the rock block. The models with face perpendicular stress demonstrate how face relaxation will precondition the rock by not only failing elements deep in the rock block, but also by causing considerable strain within the rock block. The element failure and strain near the excavation boundary is less dense than deeper into the rock block, and is likely not
enough to cause failure at the face. When the face is advanced into the preconditioned rock block, as is the case during TBM excavation, the face shows signs of instability (Figure 6.2.9). The face parallel stress does not lead to preconditioning, but its impacts on cutter-induced fracture propagation will be examined in Section 6.3.

Figure 6.2.8: Schematic of effect of stress conditions: A) face perpendicular stress; B) face parallel stress.

Figure 6.2.9: FLAC outputs of strain before ‘excavation’ (top left) and after ‘excavation’ (top right) and deformed (5x vertical magnification) grid after ‘excavation’ (bottom) showing potential instability at the face boundary. Dimensions shown in Figure 6.1.28.
6.2.2.2 Fabric Orientation with Respect to the Tunnel Face

The analysis of preconditioning for different fabric orientations shows (all output images plotted with strain at the same scale, with warmer colours representing higher strain) that preconditioning is highest for samples with fabric orientation parallel to the tunnel face (0° in Figures 6.2.10 and 6.2.11), and a general increase in preconditioning with decreasing fabric orientation to the tunnel face (Figure 6.2.10), for face perpendicular stress of 30MPa. This trend holds with a higher face perpendicular stress of 50MPa (Figure 6.2.11), although the magnitude of preconditioning is higher at this higher stress.

Figure 6.2.10: Plastic strain FLAC outputs for samples with fabric orientation (clockwise from top left) 90°, 60°, 30° and 0° to the tunnel face with stress perpendicular to the face of 30MPa.
6.2.2.3 In-Situ Stress Orientation to the Tunnel Face

Samples with fabric parallel to the tunnel face subjected to low isotropic stress of 15MPa showed failed elements (Figure 6.2.12 left) but there was no appreciable strain (Figure 6.2.12 right). Samples subjected to high isotropic stress of 50MPa showed more failed elements (Figure 6.2.13 left) and strain (Figure 6.2.13 right). Results for samples with fabric oriented at 60° to the tunnel face are presented in Figure 6.2.14.
Figure 6.2.12: FLAC output of failed elements (left) and plastic shear strain (right) for low isotropic stress with fabric at 0° to tunnel face.

Figure 6.2.13: FLAC output of failed elements (left) and plastic shear strain (right) for high isotropic stress with fabric at 0° to tunnel face.

Figure 6.2.14: FLAC output of failed elements (left) and plastic shear strain (right) for high biaxial stress with fabric at 60° to tunnel face.

The in-situ stress states outlined in Table 6.2.1 were used to investigate the magnitudes, orientations and ratios of stress that lead to enhanced preconditioning potential. Figures 6.2.15 to
6.2.17 demonstrate that increasing stress perpendicular to the tunnel face increases the level of preconditioning. Figure 6.2.17 shows that even with face parallel stress of 50MPa, no discernible preconditioning occurs, and Figure 6.2.18 shows that with constant high face perpendicular stress of 50MPa, changes in face parallel stress have no impact on preconditioning. Similar results are shown for fabric oblique to the tunnel face in Figures 6.2.19 to 6.2.22. For all tested stress conditions, the fabric parallel to the tunnel face had more preconditioning than the fabric oblique to the tunnel face.

Figure 6.2.15: Plastic strain FLAC outputs for samples with fabric parallel to tunnel face (clockwise from top left) with isotropic in-situ stress state of 10, 15, 35 and 55MPa.
Figure 6.2.16: Plastic strain FLAC outputs for samples with fabric parallel to tunnel face (clockwise from top left) with face perpendicular in-situ stress of 20, 30 and 50MPa, and face parallel in-situ stress of 12, 20 and 30.

Figure 6.2.17: Plastic strain FLAC outputs for samples with fabric parallel to tunnel face (clockwise from top left) with face perpendicular in-situ stress of 5, 15, 30 and 50MPa, and constant face parallel in-situ stress of 50MPa.
Figure 6.2.18: Plastic strain FLAC outputs for samples with fabric parallel to tunnel face (clockwise from top left) with constant face perpendicular in-situ stress 50MPa, and face parallel in-situ stress of 20, 30 and 40MPa.

Figure 6.2.19: Plastic strain FLAC outputs for samples with fabric oblique to tunnel face (clockwise from top left) with isotropic in-situ stress state of 10, 15, 35 and 55MPa.
Figure 6.2.20: Plastic strain FLAC outputs for samples with fabric oblique to tunnel face (clockwise from top left) with face perpendicular in-situ stress of 20, 30 and 50MPa, and face parallel in-situ stress of 12, 20 and 30MPa.

Figure 6.2.21: Plastic strain FLAC outputs for samples with fabric oblique to tunnel face (clockwise from top left) with face perpendicular in-situ stress of 5, 15, 30 and 50MPa, and constant face parallel in-situ stress of 50MPa.
Based on the numerical results and examination of the strain signifying preconditioning, conclusions were made regarding fabric and stress conditions that increase preconditioning, summarized in Tables 6.2.2 and 6.2.3. Magnitude of preconditioning was classified visually based on the strain output plots, all of which were plotted at the same scale with warmer colours representing greater strain.

Table 6.2.2: Summary preconditioning with respect to fabric orientation to the tunnel face.

<table>
<thead>
<tr>
<th>Fabric Orientation to Tunnel Face (Figures 6.2.10 and 6.2.11)</th>
<th>Perpendicular</th>
<th>Oblique</th>
<th>Parallel</th>
</tr>
</thead>
<tbody>
<tr>
<td>Preconditioning</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Low Preconditioning</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Moderate Preconditioning</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>High Preconditioning</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Figure 6.2.22: Plastic strain FLAC outputs for samples with fabric oblique to tunnel face (clockwise from top left) with constant face perpendicular in-situ stress 50MPa, and face parallel in-situ stress of 20, 30 and 40MPa.
Table 6.2.3: Summary of preconditioning with respect to fabric orientation to tunnel face.

<table>
<thead>
<tr>
<th>Preconditioning</th>
<th>Stress Perpendicular to Tunnel Face</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Low Stress</td>
</tr>
<tr>
<td></td>
<td>(Less than 20MPa)</td>
</tr>
<tr>
<td>Fabric Parallel to Face (Figure 6.2.18)</td>
<td>Low Preconditioning</td>
</tr>
<tr>
<td>Fabric Oblique to Face (Figure 6.2.22)</td>
<td>Low Preconditioning</td>
</tr>
<tr>
<td>Fabric Perpendicular to Face</td>
<td>Low Preconditioning</td>
</tr>
<tr>
<td>No Fabric</td>
<td>Low Preconditioning</td>
</tr>
</tbody>
</table>

### 6.2.2.4 Mineralogy, Grain Size and Fabric Type and Intensity

The different mineralogy, grain size and fabric types and intensities were investigated for their impact on preconditioning with in-situ stress conditions of 50MPa perpendicular to the tunnel face and 30MPa parallel to the tunnel face using the same sample variations used in Section 5.3. The FLAC strain outputs were used to determine relative preconditioning of mineral content (Figures 6.2.23 and 6.2.24), grain size (Figure 6.2.25), schistosity (Figure 6.2.26), cleavage (Figure 6.2.27), Mineral Preferred Orientation (MPO) and gneissic banding (Figure 6.2.28), based on the classifications in Figure 6.2.29. The samples without fabric have mild to moderate preconditioning, which is concentrated in mica grains, and increases slightly with increased mica content, but does not vary greatly with grain size. The amount of preconditioning of samples with fabric is inferred from the output plots and summarised in Table 6.2.4.

![Figure 6.2.23: FLAC strain outputs for mica variation 1 (left) and mica variation 3 (right).](image-url)
Figure 6.2.24: FLAC strain outputs for quartz variation 3 (left) and quartz variation 6 (right).

Figure 6.2.25: FLAC strain outputs for grain size 2 (left) and grain size 4 (right).
Figure 6.2.26: FLAC strain outputs for domainal schistosity (top left), Type 1 schistosity (top right) and Type 2 schistosity (bottom).

Figure 6.2.27: FLAC strain outputs for cleavage spacing >5mm (left) and cleavage spacing <5mm (right).
Figure 6.2.28: FLAC strain outputs for Mineral preferred orientation >1cm (left) and gneissic banding (right).

Table 6.2.4: Magnitude of preconditioning arising from fabric type and intensity for face perpendicular stress.

<table>
<thead>
<tr>
<th>Face Stability</th>
<th>$F_{AF}$</th>
<th>$F_{AD}$</th>
<th>Close spacing</th>
<th>Intermediate</th>
<th>Domainal</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cleavage</td>
<td></td>
<td>High</td>
<td></td>
<td>High</td>
<td></td>
</tr>
<tr>
<td>Schistosity</td>
<td></td>
<td>High</td>
<td>Moderate</td>
<td>Moderate</td>
<td></td>
</tr>
<tr>
<td>Gneissic Banding</td>
<td></td>
<td>High</td>
<td>High</td>
<td>High</td>
<td></td>
</tr>
<tr>
<td>Mineral Preferred</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Orientation</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Moderate</td>
</tr>
</tbody>
</table>

Figure 6.2.29: Schematic of $F_A$ classification scheme showing relationship between microlithon thickness divisions and foliation character and thickness used to select foliation category.
6.2.3 Field Verification of Effect of Spalling Sensitivity and Stress at the Tunnel Face on the Cutting Process

Verification of the impact of fabric orientation and in-situ stress condition on face stability was difficult with field data since information regarding orientation of the fabric was not available at the same level of detail as the mineralogy, grain size and fabric type obtained through thin section analysis. In addition, data collection was undertaken in linear tunnel sections over which very limited change in stress conditions was encountered, precluding field verification of the impact of in-situ stress condition. The samples from the Southern Aar granite (SAG) on which UCS testing was undertaken did have all of the prerequisite information necessary to fully simulate the chipping process in these rocks, using the in-situ stress condition estimate described in Section 3.2.3. Samples from the Altkristallin were also simulated under two different stress conditions to investigate the potential impact of the change of stress on an identical rock type.

6.2.3.1 Face Stability of Southern Aar Granite Rocks

The laboratory Southern Aar granite data were used to determine the impact of fabric on face instability in the SAG. A modified F-Factor specifically for face instability was quantified to obtain the best ROC curve area of 0.61 in Figure 6.2.30, suggesting that fabric is a poor predictor for face instability. The risk of each fabric type and intensity leading to face instability under high face perpendicular in-situ stress (48MPa) conditions, and reduced net advance rate based on TBM performance data from the Southern Aar granite are summarised in Table 6.2.5. The results from this investigation are similar to the results in Table 6.2.4, and are used to provide the risk designation for gneissic banding, which was not encountered in the Southern Aar granite.
Figure 6.2.30: Face instability ROC curve for fabric factor $F_A$ selected for chipping prediction

Table 6.2.5: Impact on Net Advance Rate arising from fabric type in SAG.

<table>
<thead>
<tr>
<th>Face Stability</th>
<th>$F_{AD}$</th>
<th>Close spacing</th>
<th>Intermediate</th>
<th>Domainal</th>
</tr>
</thead>
<tbody>
<tr>
<td>$F_{AF}$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cleavage</td>
<td>High Face Instability</td>
<td>Moderate Face Instability</td>
<td>High Face Instability</td>
<td></td>
</tr>
<tr>
<td>Schistosity</td>
<td>High Face Instability</td>
<td>High Face Instability</td>
<td>Moderate Face Instability</td>
<td></td>
</tr>
<tr>
<td>Mineral Preferred Orientation</td>
<td>Moderate Face Instability</td>
<td>Moderate Face Instability</td>
<td>Moderate Face Instability</td>
<td></td>
</tr>
</tbody>
</table>

6.2.3.2 Face Stability of Simulated Southern Aar Granite and Altkristallin Rocks

The stresses estimated using the methodology in Section 3.2.3 were found to lead to face instability in the Altkristallin and are, therefore, high enough to induce stress instability in foliated rock with fabric and texture similar to that encountered in the Altkristallin. The two stress conditions in Table 6.2.6 correspond to the stresses at either end of the graph in Figure 6.2.2 (700 m apart). The NAR shows a decreasing trend in this graph, suggesting increasing face instability, which, judging by the similar stress conditions in Table 6.2.6 likely arises from changes in geology as well as for changes in stress. These stresses were used in the two-cutter
model to create stress boundary conditions and investigate the impact of tunnel-induced stresses on simulated Altkristallin rock.

Images of plastic strain from the simulated Altkristallin samples (Figure 6.2.31) show that the preconditioning is not vastly different for each sample at each of the two different in-situ stress conditions, however, the preconditioning is different between the sample types. The Altkristallin sample GA_013 (Tunnel Metre 110976) was excavated under stress state 2, while sample GA_099 (Tunnel Metre 116458) was excavated under stress state 1. Both stress states have a medium face perpendicular stress, as defined in Table 6.2.3, and the fabric was oriented oblique to the tunnel face, which would suggest a moderate risk of net advance rate reduction due to preconditioning and face instability. In reality, the location from which Sample GA_099 was collected exhibited face instability (Figure 6.2.30 also reflected as low NAR in Figure 6.2.2), while the preconditioning at the location from which Sample GA_013 was collected, aided in excavation (also reflected as high NAR in Figure 6.2.2). This difference likely arises from the fabric difference, where GA_013 has intermediate cleavage and GA_099 has type 2 schistosity (intermediate). GA_099 has a high risk of NAR decrease due to the fabric type, while GA_013 has moderate risk of NAR decrease according to Table 6.2.4.

Table 6.2.6: Summary of estimated in-situ stress conditions for two-cutter modelling

<table>
<thead>
<tr>
<th>Location</th>
<th>$S1$ (MPa)</th>
<th>$S2$ (MPa)</th>
<th>$S3$ (MPa)</th>
<th>Face Parallel (MPa)</th>
<th>Face Perpendicular (MPa)</th>
<th>Out-of-plane (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>State 1</td>
<td>42</td>
<td>36</td>
<td>28</td>
<td>38</td>
<td>33</td>
<td>35</td>
</tr>
<tr>
<td>State 2</td>
<td>40</td>
<td>36</td>
<td>27</td>
<td>36</td>
<td>32</td>
<td>35</td>
</tr>
<tr>
<td>SAG</td>
<td>46-48</td>
<td>37</td>
<td>32</td>
<td>48</td>
<td>38</td>
<td>41</td>
</tr>
</tbody>
</table>
The strain results from the Southern Aar granite (Figure 6.2.32) show that the two samples without fabric (GA_b023 and GA_b088) do not show much preconditioning. The fabric of the remaining samples are intermediate cleavage and domainal schistosity, ranging in orientation from 5° to 38°, and were subjected to high face perpendicular stress (48MPa) conditions (Table 6.2.6). The face instability risk values in Tables 6.2.4 and 6.2.5 suggest that preconditioning only has a moderate risk for face instability both in terms of fabric type $F_A$ and orientation, and face perpendicular stress state. This suggests that moderate preconditioning can aid in excavation, but high preconditioning leads to a high risk of face instability.
Figure 6.2.32: FLAC strain plots from simulated Southern Aar granite samples.
The only sample that exhibited face instability was sample GA_b124 (tunnel metre 116693) while the preconditioning in all samples led to improved excavation and higher NAR (Figure 6.2.33 and also shown in Figure 6.2.1). The lack of preconditioning of sample GA_b088 (tunnel metre 116658) contributed to the poor excavation performance found in this rock type (as shown in Figure 6.2.1). Samples GA_a169, GA_b023 and GA_b039 (tunnel metres 116587, 116592 and 116608, respectively) were classified in domains E, G and H in Chapter 2, respectively, and were located in an area of frequently changing (1-2 metre scale) geological domain where the impact of each rock type is averaged over a single 2m stroke of the TBM. Samples GA_a065 and GA_b011 (tunnel metres 116484 and 116579) had moderate preconditioning leading to good excavation and high NAR, but without leading to face instability.

![Graph](image)

Figure 6.2.33: NAR-DI graph corresponding to samples used for numerical modeling, categorized according to risk of NAR reduction.
6.2.4 Summary of impact of Fabric Type, Orientation and Stress Condition on Face Stability

The analysis of factors leading to face instability, conducted using TBM performance data from the Southern Aar granite and parametric numerical modeling of fabric type and orientation, and in-situ stress condition, has resulted in the identification of potential face instability risk factors. Table 6.2.7 summarises the magnitude of preconditioning arising from fabric orientation, Table 6.2.8 summarises preliminary interpretations of the magnitude of preconditioning arising from fabric orientation and in-situ stress condition, and Table 6.2.9 summarises the potential magnitude of preconditioning arising from fabric type and intensity, with reference to Figure 6.2.34 for fabric classification types. These factors can be used in combination to estimate the potential face instability and decrease of net advance rate arising from preconditioning given the availability of fabric and in-situ stress data. The mineralogy and grain size were examined for their potential for preconditioning under high stress perpendicular to the face. The results showed that mild to moderate preconditioning occurred in rocks without fabric, with increased preconditioning with increased mica content.

Numerical modeling of samples from Altkristallin and Southern Aar granite compared to the face conditions encountered during excavation showed that rocks with low preconditioning do not seem to promote face instability. Rocks with moderate preconditioning have a low potential of inducing face instability, but the preconditioning aids the excavation and increases the drillability index, and may lead to increased net advance rate. Rocks with high preconditioning have increased potential of face instability, increasing drillability index, but also decreasing net advance rate.

Table 6.2.7: Summary of preconditioning potential with respect to fabric orientation to the tunnel face.

<table>
<thead>
<tr>
<th>Preconditioning</th>
<th>Orientation to Tunnel Face</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Perpendicular</td>
</tr>
<tr>
<td>Low Preconditioning</td>
<td>Low</td>
</tr>
<tr>
<td></td>
<td></td>
</tr>
</tbody>
</table>
Table 6.2.8: Summary of preconditioning potential with respect to fabric and in-situ stress orientation to the tunnel face.

<table>
<thead>
<tr>
<th>Preconditioning</th>
<th>Stress Perpendicular to Tunnel Face</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Low Stress (Less than 20MPa)</td>
</tr>
<tr>
<td>Fabric Parallel to Face</td>
<td>Low Preconditioning</td>
</tr>
<tr>
<td>Fabric Oblique to Face</td>
<td>Low Preconditioning</td>
</tr>
<tr>
<td>Fabric Perpendicular to Face</td>
<td>Low Preconditioning</td>
</tr>
<tr>
<td>No Fabric</td>
<td>Low Preconditioning</td>
</tr>
</tbody>
</table>

|                                 | Medium Stress (Between 20-40MPa)    |
| Fabric Parallel to Face          | Moderate Preconditioning            |
| Fabric Oblique to Face           | Moderate Preconditioning            |
| Fabric Perpendicular to Face     | Low Preconditioning                 |
| No Fabric                        | Low Preconditioning                 |

|                                 | High Stress (Greater than 40MPa)    |
| Fabric Parallel to Face          | High Preconditioning                |
| Fabric Oblique to Face           | Moderate Preconditioning            |
| Fabric Perpendicular to Face     | Moderate Preconditioning            |
| No Fabric                        | Moderate Preconditioning            |

Table 6.2.9: Summary of preconditioning potential with respect to fabric type and intensity.

<table>
<thead>
<tr>
<th>Preconditioning</th>
<th>$F_{AD}$</th>
<th>Close spacing</th>
<th>Intermediate</th>
<th>Domainal</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cleavage</td>
<td>High</td>
<td>Preconditioning</td>
<td>Moderate</td>
<td>High</td>
</tr>
<tr>
<td>Schistosity</td>
<td>High</td>
<td>Preconditioning</td>
<td>High</td>
<td>Moderate</td>
</tr>
<tr>
<td>Gneissic Banding</td>
<td>High</td>
<td>Preconditioning</td>
<td>High</td>
<td>High</td>
</tr>
<tr>
<td>Mineral Preferred Orientation</td>
<td>Moderate</td>
<td>Preconditioning</td>
<td>Moderate</td>
<td>Moderate</td>
</tr>
</tbody>
</table>

Figure 6.2.34: Schematic of $F_A$ classification scheme showing relationship between microlithon thickness divisions and foliation character and thickness used to select foliation category.
6.3 Geomechanical Characterisation for Disc Cutting

6.3.1 TBM Performance Limited by the Cutting Process

The classification of chipping performance introduced in Section 3.3 is used to determine the efficacy of the cutter excavation process. Chipping and non-chipping were terms associated with TBM performance indicators of rock that were easy or difficult to chip, respectively. Figures 6.3.1 and 6.3.2 contain TBM indicators, as per Section 3.3, from three rock masses, excavated by three comparable TBMs, with data points associated with face instability removed. The linear portion of the graph in Figure 6.3.1 is made up of strokes with non-chipping rock, while the high NAR portion of the graph is made up of strokes with chipping rock.

The data points in Figure 6.3.2 contain net thrust values on a per cutter basis to allow comparison between the different machines, each with slightly different diameters and numbers of cutters. This Figure contains the same strokes as in Figure 6.3.1 to demonstrate strokes in non-chipping (especially high thrust) and chipping (higher penetration rate) conditions, with strokes with face instability removed.

![Graph showing NAR versus net DI data from Leventina Gneiss and Southern and Central Aar Granite](image)

Figure 6.3.1: NAR versus net DI data from Leventina Gneiss and Southern and Central Aar Granite selected to highlight strokes classified as having good chipping. Data has been filtered to remove samples with face instability.
6.3.2 Numerical Investigation of Spalling Sensitivity in the Cutting Process

The two-cutter numerical model was created to investigate the impact of different geomechanical characteristics on chipping performance (Appendix F.1). The geomechanical characteristics tested in UCS and Brazilian modelling described in Chapter 5 were used in the two-cutter model. The fracture behaviour and pre-conditioning observed in the numerical models were used to make conclusions of the impact of different characteristics on chipping performance. Output images and summary of goodness of chip designations are found in Appendix F.2.

6.3.2.1 Parametric Analysis of Chipping Performance

The texture simulation algorithm developed in Appendix E.3 was used to conduct a parametric analysis of chipping performance using the two-cutter model. Each simulated rock type was input into the model. The left and right chip fracture length and depth into the rock were measured and used to determine the area affected by the chip (length x depth). The chip
area, length and depth were compared to mineralogy, grain size, and fabric type and intensity, in a similar way as the UCS and Brazilian tensile test results were compared in Chapter 5.

6.3.2.1.1 Mineralogy

The mineralogy was investigated in terms of mica content and quartz to feldspar ratio. The chip area, length and depth show a clearly positive relationship with mica content for samples with low (<0.3) quartz to feldspar ratio (Figures 6.3.3 to 6.3.5). At higher quartz to feldspar ratio (>10), no relationship is visible. These data show that rocks with mica content less than 5% and greater than 12% have higher chip areas, length and depths. This relationship is similar to the relationship between mica content and chipping performance summarized in Section 3.4.3. Very low mica will increase brittleness by providing fracture initiators that are not numerous enough to inhibit fracture propagation, high mica content will weaken the rock, in both cases having positive impacts on chipping, whereas the mica content between 4% and 12% chipping performance in reduced due to the numerous mica grains that both initiate fractures and inhibit fracture propagation.

Figure 6.3.3: Chip area versus mica content categorised according to quartz to feldspar ratio.
Figure 6.3.4: Chip length versus mica content categorised according to quartz to feldspar ratio.

Figure 6.3.5: Chip depth versus mica content (%) categorised according to quartz to feldspar ratio.
Chip area decreases slightly with increased quartz to feldspar ratio (Figures 6.3.6). For samples with low mica content, the area decreases, but the length increases with quartz to feldspar ratio (Figure 6.3.7). The depth, however, decreases with quartz to feldspar ratio (Figure 6.3.8), and although longer fracture lengths are desirable, the low fracture depth results in very thin chips, which are not as efficient for TBM excavation as thicker chips. The samples with higher quartz to feldspar ratio and low mica content form well defined, but very thin spall-like chips. The samples with mica content around 12% have a weak decrease in chip area, length and depth, with quartz to feldspar ratio, with values typically lower than for samples with lower mica content. The chip area and chip length are noticeably higher for samples with quartz to feldspar ratio lower than 1. The chipping performance summary in Section 3.4.3 suggests that 0.6 be the quartz to feldspar ratio cutoff between potential for good chipping and poor chipping, which is further supported here.

![Chip area versus quartz to feldspar ratio categorised according to mica content (%).](image)

Figure 6.3.6: Chip area versus quartz to feldspar ratio categorised according to mica content (%).
Figure 6.3.7: Chip length versus quartz to feldspar ratio categorised according to mica content (%).

Figure 6.3.8: Chip depth versus quartz to feldspar ratio categorised according to mica content (%).
6.3.2.1.2 Grain Size

The grain size analysis (Figure 6.3.9) shows that chip area and chip length, increase with increasing grain size, while chip depth decreases mildly with increasing grain size. The relationship with chip depth and grain size is not very significant since it is an artefact of the methodology by which depth is calculated: the depth of the left and right chip fracture are added. In the case of the 16mm grain size sample, the 6.5cm long 1cm deep left fracture clearly dominates the failure and leads to better chipping, but its total chip depth (with the 0.3cm deep right fracture) is low. In this case the chip area is a better measure of chipping performance, and a positive relationship between grain size and chip area, and therefore chipping performance is clearly defined.

Figure 6.3.9: Grain size versus chip dimensions: area, length and depth.
6.3.2.1.3 Fabric

The fabric analysis (Figure 6.3.10) contains several trends depending on fabric type. Only one mineral preferred orientation (MPO) sample was tested because TBM data testing in Chapter 3 showed that narrowly spaced MPO did not greatly impact chipping performance. Chip dimensions are larger for samples with schistosity rather than cleavage, likely due to the larger grain size of the microlithons defining schistosity. The chip dimensions decrease with increasing schistosity intensity due to the lack of clear weakness planes for chip fractures to exploit, while chipping dimensions increase with increasing cleavage intensity due to higher density of weakness planes along which chip fractures can propagate. Chip dimensions increase with decreasing gneissic band spacing due to closer proximity of mica-rich bands to cutters to aid in chip generation.

Figure 6.3.10: Fabric type and intensity versus chip dimensions: area, length and depth.
6.3.2.2 Summary of Chipping Performance Thresholds

Chipping performance was designated high, medium and low based on chip areas of greater than 3, 1.5 to 3 and less than 1.5 cm², respectively, obtained for different geological characteristics. The thresholds for chipping performance based on chip area for mineralogy are summarized in Tables 6.3.1 to 6.3.3.

The grain size cannot be modeled below 4mm due to element size (0.5mm) and the need to include grain size boundaries for best results. The chipping performance increases with increased grain size, as summarized in Table 6.3.4. Samples with greater than 8mm had chip areas above 7cm², which has been given the very high designation.

The fabric type and intensity combine to impact in a variety of ways without continuous trends (Table 6.3.5) due to the different impacts arising from the size and spacing of microlithons.

Table 6.3.1: Chipping performance designation for minor minerals according to accessory mineral content and which mineral makes up the greatest proportion (F_Ma).

<table>
<thead>
<tr>
<th>Chipping Performance Designation</th>
<th>Minor Minerals Content</th>
</tr>
</thead>
<tbody>
<tr>
<td>High</td>
<td>&gt;12%</td>
</tr>
<tr>
<td>Low</td>
<td>6-12%</td>
</tr>
<tr>
<td>Medium</td>
<td>2-6%</td>
</tr>
</tbody>
</table>

Table 6.3.2: Chipping performance designation for major minerals according to relative quartz content (F_MM).

<table>
<thead>
<tr>
<th>Chipping Performance Designation</th>
<th>Felsic-Mafic (Quartz to Feldspar ratio)</th>
</tr>
</thead>
<tbody>
<tr>
<td>High</td>
<td>&lt;1</td>
</tr>
<tr>
<td>Medium</td>
<td>1&lt; q/f &lt; 10</td>
</tr>
<tr>
<td>Low</td>
<td>&gt;10</td>
</tr>
</tbody>
</table>

Table 6.3.3: Combined chipping performance designation for mineralogy factor according to accessory and major mineral content (F_M).

<table>
<thead>
<tr>
<th>Chipping Performance Designation</th>
<th>F_MM</th>
<th>&lt;1</th>
<th>1&lt; q/f &lt; 10</th>
<th>&gt;10</th>
</tr>
</thead>
<tbody>
<tr>
<td>F_Ma</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>&gt;12%</td>
<td>High</td>
<td>Medium-High</td>
<td>Medium</td>
<td></td>
</tr>
<tr>
<td>6-12%</td>
<td>Medium</td>
<td>Medium-Low</td>
<td>Low</td>
<td></td>
</tr>
<tr>
<td>2-6%</td>
<td>Medium-High</td>
<td>Medium</td>
<td>Medium-Low</td>
<td></td>
</tr>
</tbody>
</table>
Table 6.3.4: Chipping performance designation for grain size ($F_{GP}$).

<table>
<thead>
<tr>
<th>Chipping Performance Designation</th>
<th>Grain Size $F_{GP}$</th>
</tr>
</thead>
<tbody>
<tr>
<td>Very High</td>
<td>&gt;8mm</td>
</tr>
<tr>
<td>High</td>
<td>4-8mm</td>
</tr>
</tbody>
</table>

Table 6.3.5: Chipping performance designation for fabric type and intensity ($F_{AD}$)

<table>
<thead>
<tr>
<th>Chipping Performance Designation</th>
<th>$F_{AD}$</th>
<th>Close spacing</th>
<th>Intermediate</th>
<th>Domainal</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cleavage</td>
<td>High</td>
<td>Low</td>
<td>Low</td>
<td>Low</td>
</tr>
<tr>
<td>Schistosity</td>
<td>Medium</td>
<td>High</td>
<td>Very High</td>
<td>Low</td>
</tr>
<tr>
<td>Banding</td>
<td>Very High</td>
<td>Medium</td>
<td>Low</td>
<td>Low</td>
</tr>
<tr>
<td>Mineral Preferred Orientation</td>
<td>Low</td>
<td>Low</td>
<td></td>
<td>Low</td>
</tr>
</tbody>
</table>

6.3.3 Numerical Investigation of Impact of Fabric Orientation and In-Situ Stress on Cutting Process

The impact of fabric orientation and in-situ stress on chipping performance was investigated by varying the fabric orientation to the tunnel face under identical applied stress conditions and by varying the stress conditions (summarised in Table 6.3.6) for two different fabric orientations.

Preconditioning of the rock due to applied stress was modelled by running the two cutter models without applying the cutters to the rock surface with results observed as failed elements and plastic strain within the rock block in Section 6.2. During the subsequent application of the cutters, the fractures at the base of the cutters were impacted by the pre-existing fractures arising from the preconditioning. Samples with fabric parallel to the cutter face showed increase in failed elements (Figure 6.3.11 left) and straining (Figure 6.3.11 right) in close proximity to the cutter. In contrast, the samples with fabric oriented 60° to the tunnel face showed similar failure and strain in close proximity to the cutters, but zones of strain developed prior to the application of the cutters are further strained after application of the cutters (Figure 6.3.13). This trend holds true for other stress conditions, and preconditioning. The cutters can take advantage of the preconditioning by using the pre-existing fractures to form larger or deeper chips, or form chips more easily or more quickly.
Table 6.3.6: Summary of in-situ stress conditions

<table>
<thead>
<tr>
<th>Stress state</th>
<th>In-Situ Stress (Input)</th>
<th>Boundary Stress (Relaxed)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Stress perpendicular to face (MPa)</td>
<td>Stress parallel to face (MPa)</td>
</tr>
<tr>
<td>Isotropic very low</td>
<td>10</td>
<td>10</td>
</tr>
<tr>
<td>Isotropic low</td>
<td>15</td>
<td>15</td>
</tr>
<tr>
<td>Isotropic medium</td>
<td>35</td>
<td>35</td>
</tr>
<tr>
<td>Isotropic high</td>
<td>55</td>
<td>55</td>
</tr>
<tr>
<td>Biaxial low</td>
<td>20</td>
<td>12</td>
</tr>
<tr>
<td>Biaxial medium</td>
<td>30</td>
<td>20</td>
</tr>
<tr>
<td>Biaxial high</td>
<td>50</td>
<td>30</td>
</tr>
<tr>
<td>Biaxial low ratio</td>
<td>50</td>
<td>40</td>
</tr>
<tr>
<td>Biaxial medium ratio</td>
<td>50</td>
<td>30</td>
</tr>
<tr>
<td>Biaxial high ratio</td>
<td>50</td>
<td>20</td>
</tr>
<tr>
<td>Biaxial Parallel high (for fabric orientation)</td>
<td>30</td>
<td>50</td>
</tr>
<tr>
<td>Biaxial Parallel medium</td>
<td>15</td>
<td>50</td>
</tr>
<tr>
<td>Biaxial Parallel low</td>
<td>5</td>
<td>50</td>
</tr>
<tr>
<td>No Damage Isotropic very low</td>
<td>0</td>
<td>10</td>
</tr>
<tr>
<td>No Damage Isotropic low</td>
<td>0</td>
<td>20</td>
</tr>
<tr>
<td>No Damage Isotropic medium</td>
<td>0</td>
<td>35</td>
</tr>
<tr>
<td>No Damage Isotropic high</td>
<td>0</td>
<td>55</td>
</tr>
<tr>
<td>No Damage Biaxial very low</td>
<td>0</td>
<td>10</td>
</tr>
<tr>
<td>No Damage Biaxial low</td>
<td>0</td>
<td>20</td>
</tr>
<tr>
<td>No Damage Biaxial medium</td>
<td>0</td>
<td>35</td>
</tr>
<tr>
<td>No Damage Biaxial high</td>
<td>0</td>
<td>55</td>
</tr>
</tbody>
</table>
Figure 6.3.11: FLAC output of failed elements (left) and plastic shear strain (right) for high isotropic stress with fabric at 0° to tunnel face after four passes of the cutter.

Figure 6.3.12: FLAC output of failed elements (left) and plastic shear strain (right) for low biaxial stress ratio with fabric at 60° to tunnel face after four passes of the cutter.

6.3.3.1 Fabric Orientation to the Tunnel Face

The impact of fabric orientation, with respect to the tunnel face, on preconditioning and chipping performance was investigated using a biaxial stress condition in which the in-situ face parallel stress was 50MPa and the perpendicular to face stress was 30MPa, which according to Section 6.2 leads to moderate preconditioning. The data in Figure 6.3.13 show that under these stress conditions, fabric oriented 60° and 30° to the face leads to the highest chip area, chip depth and chip length. The chip depth is higher for samples with fabric at 60° rather than 30° to the face due to the steeper fabric angle, but chip length is greater for samples with fabric at 60° rather than 30° to the face due to the shallower fabric angle.
When only fabric angles 60° and 0° are compared for impact of preconditioning on chip dimensions (Figure 6.3.14), chip dimensions are slightly larger under stress conditions leading to higher preconditioning, regardless of fabric orientation. For fabric oriented at 60° to the tunnel face the fracture depth is lower with high preconditioning, but the fracture length increases greatly, leading to an increase in chip area. For fabric oriented parallel to the tunnel face, the fracture length does not change with high preconditioning, but the fracture depth increases, resulting in an increase in chip area. For both fabric orientations, the fractures will follow pre-existing fractures arising from preconditioning, leading to greater localization of the induced fractures. This is especially true for fabric oriented 60° to the tunnel face, where the fracture type changes from type 2 with low preconditioning, to type 1 with high preconditioning (according to Figure F.2.1). Increased preconditioning of the tunnel face will have a greater impact on chip dimensions in tunnels in which the fabric is oriented parallel to the tunnel face.
6.3.3.2 Tunnel Boundary Stress Conditions and Fabric Orientation to the Tunnel Face

The two-cutter model was used to investigate the impact of face parallel stress on the chipping process for samples with fabric oblique to the tunnel face (45°), parallel to the tunnel face (0°) and without fabric (Figure 6.3.15). The face perpendicular stress was 0, and there was, no resulting face relaxation and no induced preconditioning. For all samples, increasing face parallel stress decreases chip area, although the greatest decrease occurred at face parallel stresses below 20MPa. For samples with fabric oblique to the face, chip depth also decreased with increasing face parallel stress, suggesting that face parallel stress will impede fracture propagation parallel and perpendicular to the face when fabric is oriented oblique to the face.

For samples with fabric parallel to the tunnel face, the chip depth did not change with stress but the chip length decreased with increasing stress due to the ease with which chips were created parallel to the narrowly spaced weakness planes of the fabric. The chip depth is, therefore, independent of face stress for samples without fabric and with fabric parallel to the tunnel face, while higher face stress, beyond 20MPa, will impede chip fracture propagation.
parallel to the tunnel face. This will lead to smaller chip area in the models, and decreased performance during TBM excavation under these conditions.

Samples with fabric oblique to the tunnel face (Figure 6.3.16) and parallel to the tunnel face (Figure 6.3.17) were run with 50MPa face perpendicular stress to create preconditioning, and the chip dimensions were compared at similar face parallel stress without preconditioning. In both cases, the samples with preconditioning had larger chip dimensions than the samples without preconditioning. The samples with oblique fabric and preconditioning show a parabolic relationship with chip dimension and face parallel stress, while the samples without preconditioning show a decreasing relationship, especially for stress less than 20MPa. Some samples without preconditioning had both isotropic stress and biaxial (out-of-plane stress = $\frac{1}{2}$ face parallel stress) stress. The dimensions for samples without preconditioning, but with biaxial face stress show slightly larger chip dimensions, showing that in either direction, face parallel stress confines the developing fractures, impeding fracture propagation.

The samples with face parallel fabric and preconditioning show a clear decrease in chip dimensions with increased face parallel stress, in particular chip depth, and resulting chip area. The chip length is increased at the highest stress, which likely arises from a pre-existing fracture due to preconditioning. The samples without preconditioning show a decrease in chip dimension, especially for stress less than 20MPa, as discussed for Figure 6.3.16.

![Figure 6.3.15: Chip dimensions compared to face boundary parallel stress condition for fabric oriented at 45° and 0° to the tunnel face and without fabric.](image-url)
Figure 6.3.16: Chip area compared to isotropic and biaxial stress condition with and without damage for fabric oriented at 60° to the tunnel face.

Figure 6.3.17: Chip dimensions compared to isotropic stress with and without damage for fabric oriented at 0° to the tunnel face.
Samples without fabric were modeled without preconditioning, and with both isotropic stress and biaxial (out-of-plane stress = \( \frac{1}{2} \) face parallel stress) stress (Figure 6.3.18). In these samples the biaxial stress leads to lower chip dimensions, in particular lower chip length. For isotropic stress, chip depth does not vary, but chip length decreases with increased stress, whereas for biaxial stress the chip length does not vary, but chip depth decreases.

Samples with fabric oblique to the tunnel face show varying relationships with face parallel stress depending on the magnitude of preconditioning (Figure 6.3.19). Samples with low preconditioning (less than 20MPa) show a decreasing chip area with increasing stress (as in Figure 6.3.16), samples with moderate preconditioning (20-40MPa) show no change in chip area at low stress but increased chip area at higher stress, although this may arise from reactivation of a deep pre-existing fracture (Figure F.2.40), and samples with high preconditioning (greater than 40MPa) show a parabolic relationship (as in Figure 6.3.17).

Samples with fabric parallel to the tunnel face show negative relationships with face parallel stress, regardless of magnitude of preconditioning (Figure 6.3.20). Samples with low to moderate preconditioning have similar chip areas, whereas samples with high preconditioning have slightly larger chip areas.

![Figure 6.3.18: Chip dimensions compared to isotropic and biaxial stress without damage for samples without fabric.](image)
Figure 6.3.19: Chip area compared to face parallel stress ratio for samples with different magnitudes of preconditioning stress and fabric oriented at 60° to the tunnel face.

Figure 6.3.20: Chip area compared to face parallel stress ratio for samples with different magnitudes of preconditioning stress and fabric oriented and 0° to the tunnel face.
6.3.3.3 Stress-Related Chip Potential Factor ($S_{\text{CP}}$)

Fabric orientation to the tunnel face and in-situ stress will impact the chipping performance in addition to the mineralogy, grain size or fabric type. As such the fabric orientation and in-situ stress condition, if available, can be used to further aid in estimating potential chipping performance. A modifier to the Geomechanical Characterisation Scheme was developed based on the two-cutter modeling results in which the impact is designated low, moderate and high according to Tables 6.3.7 and 6.3.8, where low impact is used for rocks in which the fabric orientation or stress type have no impact compared to a rock without fabric and under no in-situ stress. Medium and high are increasingly higher impacts on chipping performance inferred from the strain output plots, in that chipping performance is improved with higher impact. The preconditioning of the rock arising from pre-tunnelling face perpendicular stress found in Section 6.2.2.5 can be used to determine the impact of preconditioning, whereas the tunnel face boundary parallel stress is used to determine the impact of stress. The impact of preconditioning can either be estimated by the preconditioning stress in Table 6.3.8 or from the preconditioning potential Table 6.2.4.

Table 6.3.7: Summary of Stress-Related Chip Potential ($S_{\text{CP}}$) with respect to fabric orientation to the tunnel face.

<table>
<thead>
<tr>
<th>Stress-Related Chip Potential</th>
<th>Orientation to Tunnel Face</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Perpendicular</td>
</tr>
<tr>
<td>Low</td>
<td>Low</td>
</tr>
</tbody>
</table>

Table 6.3.8: Summary of Stress-Related Chip Potential ($S_{\text{CP}}$) with respect to fabric orientation to the tunnel face, face parallel stress and magnitude of preconditioning.

<table>
<thead>
<tr>
<th>Stress-Related Chip Potential</th>
<th>Stress Type</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Low Stress (&lt;20MPa)</td>
</tr>
<tr>
<td>Preconditioning stress</td>
<td>Low (20-40MPa)</td>
</tr>
<tr>
<td>Fabric Parallel to Face</td>
<td>Moderate</td>
</tr>
<tr>
<td>Fabric Oblique to Face</td>
<td>Moderate</td>
</tr>
<tr>
<td>Fabric Perpendicular to Face</td>
<td>Moderate</td>
</tr>
<tr>
<td>No Fabric</td>
<td>Moderate</td>
</tr>
</tbody>
</table>
6.3.4 Field Verification of Spalling Sensitivity in the Cutting Process

6.3.4.1 Comparison of Chipping Performance from Numerical Modelling and TBM Performance Data

The thresholds for chipping performance determined using numerical modeling are very similar to the thresholds determined using TBM performance data in Chapter 3. With respect to mineralogy, the thresholds are only slightly affected by changes in mica content and quartz to feldspar ratio. The results for grain size are complementary, since the TBM performance data available had very few data points with grain size greater than 5mm, and the numerical modeling data had no data points with grain size less than 4mm due to limitations in minimum element size. Where the data overlap, the trends are similar and the conclusion that chipping performance increases with increased grain size is considered valid. The methodology by which chipping performance was compared to geological characteristics in Chapter 4, while rigorous, provides less detail for threshold determination, while the methodology using chip area for numerical modelling data is considered more precise for determining thresholds because of the ability to parametrically model only one change in geological characteristics at a time.

These conclusions regarding thresholds and weightings were verified first with the modeling data, then with simulated samples from Southern Aar granite, and finally with TBM performance data collected during excavation of the Southern Aar granite. At each progressive step the weightings were adjusted, as necessary to obtain the clearest prediction results.

6.3.4.2 Quantification of F-Factors

The F-Factor weightings described as thresholds in Section 6.3.4.1 were assigned values summarized in Tables 6.3.9 to 6.3.11. The thresholds for the mineralogy factor, \( F_M \), in Table 6.3.9 are used for F-Factor calculation for chipping performance. The thresholds for the grain size factor, \( F_G \), are a combination of the factors for grain size less than 1mm from Chapter 4 and for grain size greater than 4mm from Table 6.3.4. The fabric type and intensity factor, \( F_A \), and are found in Table 6.3.11. The values for fabric orientation and stress condition are summarized in Table 6.3.12.
Table 6.3.9: Combined F-Factor values for mineralogy according to accessory and major mineral content (F_M).

<table>
<thead>
<tr>
<th>F-Factor</th>
<th>F_MM</th>
<th>1&lt;q/f&lt;10</th>
<th>1&lt;q/f&lt;10</th>
<th>&gt;10</th>
</tr>
</thead>
<tbody>
<tr>
<td>&gt;12%</td>
<td>0.7</td>
<td>0.85</td>
<td>1</td>
<td></td>
</tr>
<tr>
<td>6-12%</td>
<td>1</td>
<td>1.15</td>
<td>1.3</td>
<td></td>
</tr>
<tr>
<td>2-6%</td>
<td>0.85</td>
<td>1</td>
<td>1.15</td>
<td></td>
</tr>
</tbody>
</table>

Table 6.3.10: Combined F-Factor values for grain size and grain size designation (F_G).

<table>
<thead>
<tr>
<th>F-Factor</th>
<th>F_GS</th>
<th>&lt;0.5</th>
<th>0.5&lt;grain&lt;1</th>
<th>1&lt;grain&lt;8</th>
<th>&gt;8</th>
</tr>
</thead>
<tbody>
<tr>
<td>Isotropic</td>
<td>1.45</td>
<td>1.3</td>
<td>1.15</td>
<td>0.7</td>
<td></td>
</tr>
<tr>
<td>Seriate</td>
<td>1.3</td>
<td>1.15</td>
<td>1</td>
<td>0.55</td>
<td></td>
</tr>
<tr>
<td>Bimodal</td>
<td>1.15</td>
<td>1</td>
<td>0.85</td>
<td>0.4</td>
<td></td>
</tr>
</tbody>
</table>

Table 6.3.11: Combined F-Factor values for fabric type and intensity designation (F_A).

<table>
<thead>
<tr>
<th>Chipping Performance Designation</th>
<th>F_AF</th>
<th>Close spacing</th>
<th>Intermediate</th>
<th>Domainal</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cleavage</td>
<td>1.7</td>
<td>2</td>
<td>2</td>
<td></td>
</tr>
<tr>
<td>Schistosity</td>
<td>1.8</td>
<td>1.7</td>
<td>1.6</td>
<td></td>
</tr>
<tr>
<td>Banding</td>
<td>1.6</td>
<td>1.8</td>
<td>2</td>
<td></td>
</tr>
<tr>
<td>Mineral Preferred Orientation</td>
<td></td>
<td>2</td>
<td>2</td>
<td></td>
</tr>
<tr>
<td>No Fabric</td>
<td>1.8</td>
<td>1.8</td>
<td>1.8</td>
<td></td>
</tr>
</tbody>
</table>

Table 6.3.12: Summary of Stress-Related Chip Potential (S_{CP}) with respect to fabric orientation to the tunnel face, face parallel stress and magnitude of preconditioning.

<table>
<thead>
<tr>
<th>Stress-Related Chip Potential</th>
<th>Stress Type</th>
<th>Low Stress (&lt;20MPa)</th>
<th>Moderate (20-40MPa)</th>
<th>High Stress (&gt;40MPa)</th>
<th>Low Stress (&lt;20MPa)</th>
<th>Moderate (20-40MPa)</th>
<th>High Stress (&gt;40MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Preconditioning stress</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Fabric Parallel to Face</td>
<td>1</td>
<td>1</td>
<td>0.9</td>
<td>1.2</td>
<td>1.2</td>
<td>1</td>
<td></td>
</tr>
<tr>
<td>Fabric Oblique to Face</td>
<td>1</td>
<td>1</td>
<td>0.9</td>
<td>1</td>
<td>1</td>
<td>0.9</td>
<td></td>
</tr>
<tr>
<td>Fabric Perpendicular to Face</td>
<td>1</td>
<td></td>
<td></td>
<td>1.2</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>No Fabric</td>
<td>1</td>
<td></td>
<td></td>
<td></td>
<td>1.2</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
6.3.4.3 Calculation of Chipping Resistance Factor $C_R$

The spalling sensitivity thresholds described in Chapter 5 do not directly correlate to the chipping performance thresholds described in Chapter 4 and in Section 6.3.4.2, nor do the relationships between spalling sensitivity and chipping performance correlate to each other. For this reason, a Chipping Resistance Factor, $C_R$, has been designed to relate the geomechanical characterisation factors to chipping performance. The formula introduced in Chapter 4 for calculating Fracture Potential, $F_{F_1}$ in Equation 6.3.1 is used as the basis for the Chipping Resistance Factor $C_R$.

$$F_{F_1} = F_{SS} \times UCS = F_M \times F_G \times F_A \times \text{labstrength} \quad 6.3.1$$

In addition to the F-Factors, the Stress-Related Chipping Resistance Factor ($S_{CP}$) is used to calculate the Stress-Related Spalling Sensitivity in Equation 6.3.2 and Chipping Resistance Factor of rocks under different in-situ stress conditions according to Equation 6.3.3.

$$F_{SSA} = F_{SS} \times S_{CP} = F_M \times F_G \times F_A \times S_{CP} \quad 6.3.2$$

$$C_R = F_M \times F_G \times F_A \times \text{labstrength} \times S_{CP} \quad 6.3.3$$

6.3.4.4 Chipping Performance of Parametric Samples

The spalling sensitivity factor, $F_{SSA}$, weighted for chipping performance using the thresholds and impacts outlined in Section 6.3.4.2 (Table 6.3.13), was compared to chip area (Figure 6.3.21). With this weighting, increasing $F_{SSA}$ leads to decreasing chip area. The Chipping Resistance Factor $C_R$ was calculated according to Equation 6.3.3 using the $F_{SSA}$ values from Table 6.3.13, and UCS and Brazilian tensile strength, and was compared to chip area (Figure 6.3.22). Both graphs show similar negative relationships between $C_R$ and chip area.
Table 6.3.13: Summary of F-Factors for parametric analysis data

<table>
<thead>
<tr>
<th>Sample</th>
<th>Chip Type</th>
<th>Chip Area</th>
<th>F-Factors</th>
<th>Chipping Resistance Factor CR</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td>( F_M )</td>
<td>( F_G )</td>
</tr>
<tr>
<td>m1</td>
<td>2</td>
<td>2</td>
<td>0.85</td>
<td>0.85</td>
</tr>
<tr>
<td>m2</td>
<td>2</td>
<td>3.5</td>
<td>0.7</td>
<td>0.85</td>
</tr>
<tr>
<td>m3</td>
<td>2</td>
<td>3.9</td>
<td>0.7</td>
<td>0.85</td>
</tr>
<tr>
<td>q1</td>
<td>2</td>
<td>2</td>
<td>5</td>
<td>1</td>
</tr>
<tr>
<td>q2</td>
<td>2</td>
<td>1.8</td>
<td>1</td>
<td>0.85</td>
</tr>
<tr>
<td>q3</td>
<td>2</td>
<td>2.25</td>
<td>1</td>
<td>0.85</td>
</tr>
<tr>
<td>q36</td>
<td>2</td>
<td>0.97</td>
<td>1.15</td>
<td>0.85</td>
</tr>
<tr>
<td>q4</td>
<td>2</td>
<td>1.3</td>
<td>1</td>
<td>0.85</td>
</tr>
<tr>
<td>q6</td>
<td>2</td>
<td>1</td>
<td>1.15</td>
<td>0.85</td>
</tr>
<tr>
<td>q65</td>
<td>2</td>
<td>0.56</td>
<td>1.3</td>
<td>0.85</td>
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<tr>
<td>q7</td>
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<td>1.4</td>
<td>1.3</td>
<td>0.85</td>
</tr>
<tr>
<td>g2</td>
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<td>3.04</td>
<td>1</td>
<td>0.85</td>
</tr>
<tr>
<td>g3</td>
<td>2</td>
<td>3.4</td>
<td>1</td>
<td>0.85</td>
</tr>
<tr>
<td>g4</td>
<td>2</td>
<td>7.38</td>
<td>1</td>
<td>0.4</td>
</tr>
<tr>
<td>g5</td>
<td>2</td>
<td>7.1</td>
<td>1</td>
<td>0.4</td>
</tr>
<tr>
<td>ds</td>
<td>2</td>
<td>4.3</td>
<td>0.7</td>
<td>0.85</td>
</tr>
<tr>
<td>t1</td>
<td>2</td>
<td>2.5</td>
<td>0.7</td>
<td>1.45</td>
</tr>
<tr>
<td>t2</td>
<td>3</td>
<td>2.09</td>
<td>0.7</td>
<td>1</td>
</tr>
<tr>
<td>dc</td>
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<td>1.68</td>
<td>0.7</td>
<td>1</td>
</tr>
<tr>
<td>cc</td>
<td>2</td>
<td>2.24</td>
<td>0.7</td>
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</tr>
<tr>
<td>mpoq</td>
<td>2</td>
<td>1.75</td>
<td>1</td>
<td>0.85</td>
</tr>
</tbody>
</table>

Figure 6.3.21: Comparison between chip area and spalling sensitivity weighted according to thresholds in Section 6.3.3.2.
Figure 6.3.22: Comparison between chip area and Chipping Resistance Factor weighted according Equation 6.3.1 using UCS and Brazilian tensile strength.

The brittleness index $B_3'$, which is the product of UCS and Brazilian tensile strength, discussed in Section 6.1.4.3 had excellent correlation with spalling sensitivity of modelled UCS tests. To explore this, the Chipping Resistance Factor was calculated using the FSSA, as described above and the brittleness index $B_3'$, and was compared to chip area (Figure 6.3.23). The resulting relationship is negative with a highly pronounced sigmoidal shape. This modified Chipping Resistance Factor does not lend itself to linear regression, but rather to zonation, where $C_R$ values below 1500 result in high chipping performance, $C_R$ values between 1500 and 4000 define a transition zone resulting in medium and low chipping performance, and $C_R$ values above 4000 result in low chipping performance.

A similar pattern is visible for the data with Fracture Potential (Figure 6.3.24) calculated according to Equation 6.3.2, with UCS and BTS, and weighted according to spalling sensitivity of UCS tests, discussed in Section 6.1.4.3. Although the thresholds determined in Chapter 5 and weightings from Section 6.1.4.3 do not completely match the thresholds from Chapter 4 and
Figure 6.3.23: Comparison between chip area and Chipping Resistance Factor weighted according chipping performance calculated using Equation 6.3.3 using UCS and Brazilian tensile strength.

Figure 6.3.24: Comparison between chip area and modified Fracture Potential weighted according spalling sensitivity of UCS test calculated using Equation 6.3.1 using UCS and Brazilian tensile strength.
weightings from Section 6.3.3.2, the resulting Fracture Potential and Chipping Resistance Factor versus chip area graphs are very similar. A threshold can be selected based on either weighting system from which chipping performance is divided into good or poor, although the thresholds for chipping performance appear better defined using the Chipping Resistance Factor $C_R$ than the Fracture Potential.

### 6.3.4.5 Chipping Performance of Simulated Southern Aar Granite Rocks

The fabric orientation in the Southern Aar granite was in general between 5° and 38° to the tunnel face and thus falls between the oblique and parallel fabric orientation designations in Table 6.3.12. In addition, the stress condition under which the Southern Aar granite was excavated had high pre-tunnelling tunnel face perpendicular and parallel stress (nearly hydrostatic), which results in high impact on chipping performance for fabric oriented oblique and parallel to the tunnel face, respectively. Field verification of the impact of fabric orientation and in-situ stress was not possible because the fabric orientation was not specifically recorded for each sample in the Southern Aar granite and the stress condition did not change considerably over the tunnel length from which the samples were collected. Field verification can only be done using the five Southern Aar granite samples for which fabric orientation data are available. The Southern Aar granite rocks simulated in Section 5.2 were used in the two-cutter model to compare the model behaviour with the behaviour interpreted from TBM performance data, as outlined in Section 3.3.

Chip dimensions were found to decrease with increasing laboratory UCS (Figure 6.3.25) and modelled UCS (Figure 6.3.26). These results are based on trends, but variability is quite high, showing that UCS alone is insufficient to predict chipping performance. Figure 6.3.27 shows that $C_R$ corresponds well to TBM performance classified with a NAR versus DI graph. Sample GA_a065 does not fit well since it was exhibited some preconditioning leading to moderate instability.
Figure 6.3.25: Comparison of chip dimensions to laboratory UCS values for Southern Aar granite samples.

Figure 6.3.26: Comparison of chip dimensions to modelled UCS values for Southern Aar granite samples.
6.3.4.6 Chipping Performance of Southern Aar Granite Rocks

6.3.4.6.1 Samples without Fabric

The samples of Southern Aar granite used for chipping performance analysis in Chapter 4 were also used to investigate the ability of the spalling sensitivity weighted according to the thresholds in Section 6.3.3.2 to predict chipping performance. The spalling sensitivity compared to the net drillability index (Figure 6.3.28) produces a negative relationship. The relationship is more clearly defined when the spalling sensitivity is used to calculate the Chipping Resistance Factor using the face perpendicular and parallel point load index strength and Equation 6.3.3 (Figure 6.3.29). The relationship with PLT parallel is similar to the relationships in Figures 6.3.22 to 6.3.24, while the relationship with PLT perpendicular is inconclusive. Using the PLT parallel, the samples with Chipping Resistance Factor above 4000 have net DI below 0.05mm/rev.kN while the samples with $C_R$ 4000 and lower have net DI above 0.05mm/rev.kN.
Figure 6.3.28: Comparison between drillability index and spalling sensitivity weighted according to thresholds in Section 6.3.3.2.

Figure 6.3.29: Comparison between drillability index and Chipping Resistance Factor weighted according to Equation 6.3.1, using point load index strength perpendicular parallel to tunnel face.
The DI versus $C_R$ graph in Figure 6.3.30 was categorized by chipping performance and face instability (Figure 6.3.31) to highlight the applicability of $C_R$ to predict chipping performance in terms of DI, as well as those points that do not perfectly fit the system. Samples for which face instability was encountered tend to have low $C_R$, although one point was designated with a high $C_R$. The $C_R$ was also plotted versus net advance rate (Figure 6.3.32) to highlight the applicability of $C_R$ to predict chipping performance in terms of NAR. All $C_R$ values 4000 and lower, have NAR values above approximately 30mm/min, while most $C_R$ values above 4000, have NAR values below approximately 30mm/min. A few points with $C_R$ values above 4000 have NAR values above 30mm/min. These points are shown on the NAR versus DI graph (Figure 6.3.32) relating to the samples in Figures 6.3.30 and 6.3.31. Two points are on the boundary between poor chipping and good chipping, while the third is well within the good chipping zone, but near the boundary between face stability and face instability. These points are special cases and outliers showing that misclassification is possible, however, with the misclassification being conservative, where samples were classified as having low chipping, but in reality exhibited high chipping.

![Figure 6.3.30: Comparison between drillability index and Chipping Resistance Factor weighted according to Equation 6.3.1, using point load index strength parallel to tunnel face, categorized according to chipping performance and face stability.](image)
Figure 6.3.31: Comparison between net advance rate and Chipping Resistance Factor weighted according to Equation 6.3.1, using point load index strength parallel to tunnel face, categorized according to chipping performance and face stability.

Figure 6.3.32: NAR versus DI graph for Southern Aar granite samples without fabric, categorized according to chipping performance and face stability.
6.3.4.6.2 Samples with Fabric

To investigate the impact of fabric type and intensity on chipping performance, values were assigned an FA value and were weighted to obtain the highest ROC area. The resulting ROC curve has an area of 0.69 (Figure 6.3.33), which suggests that fabric, weighted in this configuration is a fair predictor of chipping performance. The weightings from this investigation compare well with the weightings from the numerical modelling analysis.

![ROC Curve](image)

Figure 6.3.33: Chipping performance ROC curve for fabric factor FA selected for chipping prediction.

6.3.4.6.3 Verification of the Chipping Resistance Factor CR in Southern Aar Granite Rocks

The Chipping Resistance Factor was applied to TBM performance data from the Southern Aar granite to verify that the system could predict poor chipping. The approach to calculating CR was modified for the data from the SAG, since UCS and Brazilian tensile strength data were not available. Point load index strength data were available, however, and a constant of value 750, was multiplied by the CR value based on PLT strength, to make it comparable to CR based on UCS and Brazilian tensile strength. Since the rocks investigated were all excavated under very similar fabric orientation and stress conditions, the value for SCP was 0.9 (oblique orientation with moderate preconditioning and high face parallel stress) for all samples. No
values for chip area were available for the SAG samples, but Drillability Index (DI) was used to represent chipping performance. Figure 6.3.34 shows the Chipping Resistance Factor with PLT parallel compared to DI with a sigmoidal relationship. A similar analysis was undertaken with Net Advance Rate with a similar sigmoidal relationship (Figure 6.3.35). Thresholds have been identified coincident with the thresholds in 6.3.23 designating the risk of reduced TBM performance (NAR and DI).

![Figure 6.3.34: Comparison between drillability index and Chipping Resistance Factor weighted according to Equation 6.3.1, using point load index strength parallel to tunnel face, categorized according to chipping performance and face stability.](image)

Figure 6.3.34: Comparison between drillability index and Chipping Resistance Factor weighted according to Equation 6.3.1, using point load index strength parallel to tunnel face, categorized according to chipping performance and face stability.
Figure 6.3.35: Comparison between net advance rate and Chipping Resistance Factor weighted according to Equation 6.3.1, using point load index strength parallel to tunnel face, categorized according to chipping performance and face stability.

Figure 6.3.36 shows the NAR versus DI graph, with the points categorised according to their $C_R$ value, showing that the $C_R$ can be used to predict risk of poor chipping performance (circled). The prediction can be conservative, demonstrated by points with $C_R > 1500$ in good-chipping conditions, but only one point with $C_R < 1500$ actually exhibited poor chipping conditions. The $C_R$ values were used to categorise the start-up tests discussed in Chapter 3 in a penetration-thrust graph (Figure 6.3.37). The distribution of the categorised points fit the chipping performance thresholds determined in Chapter 3. As with Figure 6.3.36, this shows that the $C_R$ value can be used to predict chipping performance, again somewhat conservatively.
Figure 6.3.36: NAR-DI graph corresponding to Southern Aar granite samples categorized according to Chipping Resistance Factor calculated using the point load index strength.

Figure 6.3.37: Penetration versus thrust graph of Start-up test data, categorized according to the $C_R$ value, where white is $<1500$, grey is $1500-4000$ and black is $>4000$. 
6.3.5 Summary of Chipping Resistance Factor and Stress-Related Chip Potential

The two-cutter numerical modeling of chipping performance was used to determine thresholds for high, medium and low chip area. These designations were compared to the thresholds for chipping performance determined using Southern Aar granite samples and TBM performance indicators. The thresholds were very similar, and because the numerical modeling analysis provided more precise thresholds, these were used for chipping specific determination of the mineralogy factor, $F_m$, the grain size factor, $F_g$, and the Fabric factor $F_A$. These factors were used to evaluate the spalling sensitivity, $F_{SSA}$, weighted for chipping performance. The spalling sensitivity was then compared to chip area of numerical model results, as well as net DI and NAR of Southern Aar granite samples. The trends were very similar but not clear enough for use as quantified predictor formulas. The Chipping Resistance Factor, $C_R$, was calculated in a similar fashion as the Fracture Potential, using the spalling sensitivity and different laboratory strength values. The Modified $C_R$ was calculated using the product of the spalling sensitivity, UCS and Brazilian tensile strength, producing the clearest trend for numerical modeling data. The Southern Aar granite data were tested for point load index strength perpendicular and parallel to the tunnel face. These values were used to calculate the $C_R$, and the $C_R$ calculated using parallel PLT provided the clearest trend. The $C_R$ was also compared to net advance rate, again with a clear trend. The resulting $C_R$ thresholds are summarized in Table 6.3.14.

Table 6.3.14: Summary of thresholds for Chipping Resistance Factors

<table>
<thead>
<tr>
<th>Chipping Resistance Factor Type</th>
<th>Low Risk of Poor Chipping</th>
<th>Moderate Risk of Poor Chipping</th>
<th>High Risk of Poor Chipping</th>
</tr>
</thead>
<tbody>
<tr>
<td>$F_{SSA} = F_A \times F_G \times F_M \times S_{CP}$</td>
<td>$&lt;1.3$</td>
<td>$1.3-1.5$</td>
<td>$&gt;1.5$</td>
</tr>
<tr>
<td>$C_R = F_{SSA} \times UCS \times BTS$</td>
<td>$&lt;1500$</td>
<td>$1500-4000$</td>
<td>$&gt;4000$</td>
</tr>
<tr>
<td>$C_R = 750 \times F_{SSA} \times PLT_d$</td>
<td>$&lt;1500$</td>
<td>$1500-4000$</td>
<td>$&gt;4000$</td>
</tr>
</tbody>
</table>
6.4 Geomechanical Characterisation Scheme

The Geomechanical Characterisation Scheme developed in Chapter 4 was modified in Sections 6.1 to 6.3 through calibration and verification, using TBM excavation and numerical modelling data. The Chipping Resistance Factor $C_R$ is the main output value from the Geomechanical Characterisation Scheme (Figure 6.4.1) and the face preconditioning is the main output from estimates of in-situ stress conditions and fabric type and orientation. These two can be used to estimate the TBM excavation performance with respect to penetration rate and required thrust (Figure 6.4.2) and/or drillability index and net advance rate according to Figure 6.4.3. The penetration-thrust graph, introduced in Chapter 3, demonstrates the relationship between applied TBM thrust and the resulting penetration depth for each cutterhead revolution, and the impact rock strength has on both factors. The NAR-DI graph, also introduced in Chapter 3, demonstrates the relationship between the drillability index (penetration rate normalized by thrust) and the net advance rate, and can be used to determine decreased performance arising from tough rock conditions or stress-induced face instability. The key indicator is net advance rate, where high advance rates (>30mm/min in the SAG) are most desirable and can be reduced either by poor chipping (estimated using $C_R$) or by face instability (estimated using $A_{FS}$ and fabric type and intensity).

![Flowchart demonstrating the Geomechanical Characterisation Scheme.](image)

Figure 6.4.1: Flowchart demonstrating the Geomechanical Characterisation Scheme.
6.4.1 Geomechanical Characterisation at Disc Cutter Scale

Based on the TBM performance analysis, numerical calibration and field verification, thresholds and values were associated with the geological F-Factors that were identified for their impact on the chipping process and face stability. The Geomechanical Characterisation Scheme was defined, in which the mineralogy (FM Table 6.4.1), grain size and grain size distribution (FG Table 6.4.2), and fabric type and intensity (FA Table 6.4.3) were weighted according to their impact on the chipping process. Fabric orientation and in-situ stress condition were investigated for their impact on the Stress-Related Chipping Potential (SCP Table 6.4.4). The F-Factors identified for the geomechanical scheme are combined to determine the spalling sensitivity, which, combined with laboratory strength data and Stress-Related Chipping Potential, can be used to determine the Fracture Potential or Chipping Resistance Factor (CR).

Table 6.4.1: Combined F-Factor values for mineralogy according to accessory and major minerals (FM).

<table>
<thead>
<tr>
<th>F-Factor</th>
<th>FM&gt;M</th>
<th>&lt;1</th>
<th>1&lt; q/f &lt; 10</th>
<th>&gt;10</th>
</tr>
</thead>
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<tr>
<td>&gt;12%</td>
<td>0.7</td>
<td>0.85</td>
<td>1</td>
<td></td>
</tr>
<tr>
<td>6-12%</td>
<td>1</td>
<td>1.15</td>
<td>1.3</td>
<td></td>
</tr>
<tr>
<td>2-6%</td>
<td>0.85</td>
<td>1</td>
<td>1.15</td>
<td></td>
</tr>
</tbody>
</table>

Table 6.4.2: Combined F-Factor values for grain size and grain size distribution (FG).

<table>
<thead>
<tr>
<th>F-Factor</th>
<th>FG&gt;S</th>
<th>&lt;0.5</th>
<th>0.5&lt; grain &lt; 1</th>
<th>1&lt; grain &lt; 8</th>
<th>&gt;8</th>
</tr>
</thead>
<tbody>
<tr>
<td>Isotropic</td>
<td>1.45</td>
<td>1.3</td>
<td>1.15</td>
<td>0.7</td>
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<tr>
<td>Seriate</td>
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<td>1.15</td>
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<td>0.55</td>
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<td>1</td>
<td>0.85</td>
<td>0.4</td>
<td></td>
</tr>
</tbody>
</table>

Table 6.4.3: Combined F-Factor values for fabric type and intensity designation (FA).

<table>
<thead>
<tr>
<th>Chipping Performance Designation</th>
<th>FA&gt;D</th>
<th>Close spacing</th>
<th>Intermediate</th>
<th>Domainal</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cleavage</td>
<td>1.7</td>
<td>2</td>
<td>2</td>
<td></td>
</tr>
<tr>
<td>Schistosity</td>
<td>1.8</td>
<td>1.7</td>
<td>1.6</td>
<td></td>
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<tr>
<td>Gneissic Banding</td>
<td>1.6</td>
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<td>Mineral Preferred Orientation</td>
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<td>2</td>
<td></td>
</tr>
<tr>
<td>No Fabric</td>
<td>2</td>
<td>2</td>
<td>2</td>
<td></td>
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</tbody>
</table>
### Table 6.4.4: Summary of values for Stress-Related Chip Potential Factor ($S_{CP}$)

<table>
<thead>
<tr>
<th>Stress-Related Chip Potential</th>
<th>Stress Type</th>
<th>Low Stress (&lt;20MPa)</th>
<th>Moderate (20-40MPa)</th>
<th>High (&gt;40MPa)</th>
<th>Low Stress (&lt;20MPa)</th>
<th>Moderate (20-40MPa)</th>
<th>High (&gt;40MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Preconditioning</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Fabric Parallel to Face</td>
<td>Low (≤20MPa)</td>
<td>1</td>
<td>1</td>
<td>0.9</td>
<td>1.2</td>
<td>1.2</td>
<td>1</td>
</tr>
<tr>
<td>Fabric Oblique to Face</td>
<td>Low (≤20MPa)</td>
<td>1</td>
<td>1</td>
<td>0.9</td>
<td>1</td>
<td>1</td>
<td>0.9</td>
</tr>
<tr>
<td>Fabric Perpendicular to Face</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>No Fabric</td>
<td></td>
<td>1</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>1.2</td>
</tr>
</tbody>
</table>

The Chipping Resistance Factor is calculated according to Figure 6.4.1 and:

$$C_P = F_M \times F_G \times F_A \times labstrength \times S_{CP}$$  \hspace{1cm} 6.4.1

Thresholds were identified for the Chipping Resistance Factor based on laboratory strength values from UCS, UCS and Brazilian, or point load testing beyond which the risk of poor chipping conditions is elevated (Table 6.4.5). The Chipping Resistance Factor was calculated for the Southern Aar granite to verify that it is capable of identifying geological conditions that exhibited poor chipping during TBM excavation (Figures 6.4.2 and 6.4.3). The methodology was able to identify samples for which poor chipping conditions were encountered. The misplaced points demonstrate that in some cases the Chipping Resistance Factor tends to conservatively estimate the risk of poor chipping. The individual F-Factor values and resulting $C_R$ values for all samples are found in Appendix F.3 compared to TBM performance and tunnel geology and wall stability maps.

### Table 6.4.5: Summary of thresholds for Chipping Resistance Factors

<table>
<thead>
<tr>
<th>Chipping Resistance Factor Type</th>
<th>Low Risk of Poor Chipping</th>
<th>Moderate Risk of Poor Chipping</th>
<th>High Risk of Poor Chipping</th>
</tr>
</thead>
<tbody>
<tr>
<td>$F_{SSA} = F_A \times F_G \times F_M \times S_{CP}$</td>
<td>&lt;1.3</td>
<td>1.3-1.5</td>
<td>&gt;1.5</td>
</tr>
<tr>
<td>$C_R = F_{SSA} \times UCS \times BTS$</td>
<td>&lt;1500</td>
<td>1500-4000</td>
<td>&gt;4000</td>
</tr>
<tr>
<td>$C_R = 750 \times F_{SSA} \times PLT_d$</td>
<td>&lt;1500</td>
<td>1500-4000</td>
<td>&gt;4000</td>
</tr>
</tbody>
</table>
Figure 6.4.2: NAR-DI graph corresponding to Southern Aar granite samples categorized according to Chipping Resistance Factor calculated using the point load index strength.

Figure 6.4.3: Penetration versus thrust graph of Start-up test data, categorized according to the $C_R$ value, where white is $<1500$, grey is 1500-4000 and black is $>4000$. 
6.4.2 Geomechanical Classification at Tunnel Scale

The analysis of factors leading to face instability conducted using TBM performance data from the Southern Aar granite and parametric numerical modeling of fabric orientation and in-situ stress condition has resulted in identification of face instability risk factors. Table 6.4.6 summarises the magnitude of preconditioning arising from fabric orientation, Table 6.4.7 summarises the magnitude of preconditioning arising from fabric orientation and in-situ stress condition, and Table 6.4.8 summarises the risk factors arising from fabric type and intensity.

The fabric type and intensity are determined according to the schematic in Figure 6.4.3. Rocks with low preconditioning do not lead to face instability, and have no impact on drillability index. Rocks with moderate preconditioning have a low risk of creating face instability, but the preconditioning aids the excavation and increases the drillability index, and may lead to increased net advance rate. Rocks with high preconditioning have increased risk of face instability, increasing drillability index, but also decreasing net advance rate.

Table 6.4.6: Summary preconditioning with respect to fabric orientation to the tunnel face.

<table>
<thead>
<tr>
<th>Orientation to Tunnel Face</th>
<th>Perpendicular</th>
<th>Oblique</th>
<th>Parallel</th>
</tr>
</thead>
<tbody>
<tr>
<td>Preconditioning</td>
<td>Low Preconditioning</td>
<td>Moderate Preconditioning</td>
<td>High Preconditioning</td>
</tr>
</tbody>
</table>

Table 6.4.7: Summary of preconditioning with respect to fabric orientation to the tunnel face.

<table>
<thead>
<tr>
<th>Preconditioning</th>
<th>Stress Perpendicular to Tunnel Face</th>
<th>Low Stress (Less than 20MPa)</th>
<th>Medium Stress (Between 20-40MPa)</th>
<th>High Stress (Greater than 40MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fabric Parallel to Face</td>
<td>Low Preconditioning</td>
<td>Moderate Preconditioning</td>
<td>High Preconditioning</td>
<td></td>
</tr>
<tr>
<td>Fabric Oblique to Face</td>
<td>Low Preconditioning</td>
<td>Moderate Preconditioning</td>
<td>Moderate Preconditioning</td>
<td></td>
</tr>
<tr>
<td>Fabric Perpendicular to Face</td>
<td>Low Preconditioning</td>
<td>Low Preconditioning</td>
<td>Moderate Preconditioning</td>
<td></td>
</tr>
<tr>
<td>No Fabric</td>
<td>Low Preconditioning</td>
<td>Low Preconditioning</td>
<td>Moderate Preconditioning</td>
<td></td>
</tr>
</tbody>
</table>
Table 6.4.8: Summary of preconditioning with respect to fabric type and intensity.

<table>
<thead>
<tr>
<th>Preconditioning</th>
<th>$F_{AD}$</th>
<th>Close spacing</th>
<th>Intermediate</th>
<th>Domainal</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>$F_{AF}$</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cleavage</td>
<td>High</td>
<td>Moderate</td>
<td>High</td>
<td>Preconditioning</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Preconditioning</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Schistososity</td>
<td>High</td>
<td>High</td>
<td>Moderate</td>
<td>Preconditioning</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Preconditioning</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Gneissic Banding</td>
<td>High</td>
<td>High</td>
<td>High</td>
<td>Preconditioning</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Preconditioning</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Mineral Preferred</td>
<td>Moderate</td>
<td>Moderate</td>
<td>Moderate</td>
<td>Preconditioning</td>
</tr>
<tr>
<td>Orientation</td>
<td></td>
<td>Preconditioning</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

The risk of NAR reduction arising from face instability was assessed for the Southern Aar granite samples, for which stress and fabric orientation information were available, to verify that it is capable of identifying geological conditions that exhibited NAR reduction during TBM excavation (Figure 6.4.4). The methodology was able to identify the single sample with face instability, all samples with moderate instability, and all samples with no face instability conditions encountered. The single misplaced point demonstrates that in some cases the risk of NAR reduction estimate tends to be conservative.

Figure 6.4.4: NAR-DI graph corresponding to samples used for numerical modeling, categorized according to risk of NAR reduction.
6.4.3 Application of Geomechanical Characterisation Scheme to a Suite of Rocks

The Geomechanical Characterisation Scheme was used to estimate Spalling Sensitivity $F_{SSA}$ for samples collected from the Altkristallin, Leventina gneiss and Central Aar granite. For each rock type, several samples were characterised using thin sections, although no laboratory strength testing was undertaken. The stress condition in the Altkristallin was estimated in Section 6.1.5.2, although no stress estimates were possible for the Leventina gneiss or the Central Aar granite. The Central Aar granite samples did not have any fabric, and the Stress-Related Chipping Potential $S_{CP}$ was 1, while the Leventina gneiss has fabric oriented roughly perpendicular to the tunnel face, making the $S_{CP}$ also 1. With this information, $F_{SSA}$ was calculated according to:

$$F_{SSA} = F_M \times F_G \times F_A \times S_{CP}$$

The resulting data were used to categorise the sample data according to Low, Medium and High $F_{SSA}$ values with the following thresholds:

- Low $F_{SSA} = <1.3$
- Medium $F_{SSA} = 1.3 < F_{SSA} < 1.5$
- High $F_{SSA} = >1.5$

![NAR-DI graph](image)

Figure 6.4.5: NAR-DI graph corresponding to Altkristallin, Leventina gneiss and Central Aar granite samples categorized according to Chipping Resistance Factor calculated without laboratory strength values.

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These data were used to plot NAR and DI data in Figure 6.4.5. The zone highlighted as poor chipping, according to Figure 6.4.5 contains only points with high F_{SSA} values. There are high F_{SSA} points in good chipping parts of the NAR versus DI graph, but there are no medium or low F_{SSA} points, except for one low F_{SSA} point from the Leventina gneiss. The key is to identify rock types at risk of poor chipping conditions without mischaracterizing rocks as high chipping that actually exhibit poor chipping during excavation. This analysis shows that the Geomechanical Characterisation Scheme is capable of doing this, albeit fairly conservatively. This analysis also shows that the Geomechanical Characterisation Scheme developed using Southern Aar granite and numerical modelling data can be applied to other rock types.

The impact of magnitude of preconditioning on face instability was tested on other rock types, the Leventina gneiss and Central Aar granite, regardless of lack of specific stress estimates and fabric orientation data. These estimates were used to classify the NAR and DI data for Altkristallin, Leventina gneiss and Central Aar granite in a NAR versus DI graph (Figure 6.4.6). The results show that the estimates were able to predict locations with face instability for the Altkristallin (the Leventina gneiss had no face instability, nor was any predicted) but not for the Central Aar granite. More specific stress and fabric orientation information would be necessary to reliably predict risk of NAR reduction arising from face instability in the Central Aar granite.

![NAR-DI graph](image)

Figure 6.4.6: NAR-DI graph corresponding to Altkristallin, Leventina gneiss and Central Aar granite samples categorized according to Chipping Resistance Factor calculated without laboratory strength values.
Chapter 7 Summary, Conclusions and Recommendations

7.1 Summary

This thesis presents an exploration of the important impact of geological characteristics on massive rock yield at the tunnel face in deep underground hard rock excavations. This study was achieved by combining geological and TBM performance data collected in three tunnels excavated by tunnel boring machines, published literature regarding the relationships between geological characteristics and rock yielding processes, and numerical modeling of laboratory strength tests and the TBM cutting process.

7.1.1 TBM Excavation of the Swiss Alpine Tunnels

Efficient TBM penetration relies on ensuring that a chipping mechanism dominates over a crushing, grinding, or plastic deformation mechanism under the cutter. The chipping hypothesis employed throughout this research postulates that chipping is accomplished by inducing tensile fractures into and parallel to the tunnel face. The fractures induced in the rock act to precondition the rock, making it easier to excavate during subsequent rotations of the cutterhead. The fractures induced parallel to the tunnel face will coalesce with similar fractures induced by adjacent cutters to create chips. It is generally agreed that chip creation is the most efficient process for rock excavation, while the least efficient excavation process is grinding, which is postulated to occur at the cutter-rock interface.

TBM performance data were collected using a data acquisition system built into the Herrenknecht TBM control system. The key parameters that were used in analysis were the penetration rate, gross thrust, torque and RPM. In order to obtain a detailed dataset of parameters over the full range of penetration and thrust, start-up tests were developed in which the TBM operator slowly increased the TBM thrust after a full stop. In addition to TBM performance data, geological data were collected in the form of rock samples, tunnel wall and face maps, photos and detailed descriptions, rock mass characterisation and rock core point load strength testing.

The TBM performance data were processed to obtain average values assigned to each TBM stroke. The TBM performance data were also processed such that they could be geographically related to the geological and strength testing data according to the location along the tunnel length at which the data were collected. This allowed the data to be analysed in terms...
of the TBM performance data, and compared to the geological conditions under which they were collected.

The TBM performance data were analysed to determine the impact of the intact rock conditions at the face as well as stress induced failure. A process at the tunnel face was described in which the interaction of the induced stress condition, the intact rock strength, spalling sensitivity and fabric type and orientation led to induced face instability through originally massive rock. The net advance rate was developed to identify the strokes at which face instability occurred. The net advance rate is calculated by dividing the stroke length by the active driving time, which is the total time during which the TBM head was actively turning. This ratio takes into account the negative impact on advance rate, due to the need to remove fallen unstable tunnel face material during tunnelling, by TBM cutterhead rotation without applied thrust. It considers the full range of possible tunnelling conditions, including tough rock, preconditioned rock and rock undergoing face instability.

The analysis and quantification of start-up test data, strength testing data and average stroke running data were used to develop a methodology by which chipping performance and induced face instability could be categorised. The penetration – thrust graph was used to determine the chipping performance category while the drillability index – NAR graph was used to confirm the chipping performance category and determine the face stability category. The thresholds in these graphs were verified by categorisation of the start-up test data. This methodology was used to categorize the location at which each rock sample used for geological analysis was collected into chipping versus non-chipping and stable versus unstable tunnel face.

Using the drill core samples and tunnel wall map records, the geology along a 400m section of the Southern Aar granite was classified into dimensionless domains using the following criteria: mineralogical components, median grain size, grain size distribution, fabric type and overall variability, in terms of shear zones, fractures, rock type and tunnel wall overbreak. The PLT index strength test data, geological domains and TBM performance domains were used to investigate their ability to differentiate chipping performance and tunnel face stability categories. It was found that PLT index strength test data were an excellent predictor of chipping performance, in which rocks with face perpendicular strength below 6MPa and face parallel strength below 2MPa exhibited good chipping performance. The diametral strength was a better predictor due to the orientation of the induced fracture parallel to the fracture orientation during chipping.
7.1.2 Micromechanics and Rock Behaviour

A literature review was conducted to determine the geological factors that were most important in controlling the failure behaviour of intact rocks. Three types of factors were identified: the mineralogy, grain size and grain size distribution, and the fabric type and intensity. These factors were combined into a geomechanical characterisation scheme that results in a quantification of the spalling sensitivity and fracture potential.

- The mineralogy was categorised according to major (i.e. quartz, olivine, feldspar, calcite, amphibole, and pyroxene) and accessory (i.e. biotite, muscovite, garnet, pyrite and magnetite) minerals. Each category uses the total and relative mineral contents, which are incorporated into the mineralogy factor $F_M$.

- The grain size and grain size distribution were categorised according to three grain size ranges and three grain size distribution types (isotropic, seriate and bimodal). The categories are incorporated into the grain size and grain size distribution factor $F_G$.

- The fabric type and intensity were categorised into four fabric types (mineral preferred orientation, schistosity, cleavage and gneissic banding) and three intensity categories based on microlithon spacing.

The F Factors were developed as part of the overall methodology to relate geological characteristics to rock mechanics demonstrated. The term spall sensitivity, $F_{SS}$, is used to describe the impact that mineralogy and grain size have on rock yield behaviour that could lead to sudden spalling at the excavation boundary. The Stress-Related Spall Sensitivity, $F_{SSA}$, includes the additional impact of intensity of foliation for anisotropic rocks. The spall sensitivity factor, combined with standard lab strength predictions, describes differences in rock yield behaviour under induced stress conditions during TBM excavation. For this purpose, $F_{RI}$ is used to describe the potential for fracture at the cutter scale and the entire tunnel face scale in isotropic rock. For rocks with geological fabric anisotropy (not anisotropy due to fracture sets) the Stress-Related Spall Sensitivity factor is used and the result is referred to as the Stress-Related fracture potential, $F_{FA}$.

This methodology can be used to anticipate rock yield behaviour at excavation boundaries to make better predictions for TBM performance. This system was created for crystalline rocks, not sedimentary rocks, due to the difference in characteristics of grain boundaries found in sedimentary rocks. Ultramafic, volcanic and highly altered ore rocks are also not considered due to limited data for these rock types.
The TBM performance data were used to categorize each rock sample characterized by the geomechanical characterisation scheme according to chipping performance and tunnel face stability. The geomechanical characterisation scheme was assessed using the chipping performance and tunnel face stability categories assigned to the rock samples. The very few samples that coincided with tunnel face instability contain fabric, and as such the mineralogy, $F_m$, and grain size and grain size distribution, $F_G$, could not be evaluated for their impact on tunnel face stability. The analysis of the impact of fabric on tunnel face stability showed that the designations for high, medium and low face instability vary roughly with intensity, in different ways depending on fabric type.

### 7.1.3 Numerical Calibration of Geomechanical Characterisation Scheme

A series of numerical models were created to simulate laboratory strength tests in two dimensions. In order to perform a parametric analysis of the geological factors comprising the geomechanical characterisation scheme using these strength test models a methodology was developed to explicitly simulate rock textures and geological characteristics at the grain size scale. This methodology consists of separate constitutive models for mica, quartz and feldspar, as well as a method by which each mineral type is simulated in a realistic rock aggregate analogue based on thin sections of rocks from the Gotthard tunnel.

Rock analogues were simulated by the generation of mineral aggregates using simple geometrical algorithms for generating ellipses. This methodology allows explicit control on aggregate size and orientation through specification of ellipse size and elongate axis orientation. In addition, grain boundaries can also be explicitly generated. Individual mineral types, with appropriate shapes and grain boundaries are explicitly generated and assigned mineral-specific properties.

A series of 24 sample types, each with different geological characteristics, in terms of mineralogy, grain size and fabric type, were used for parametric analysis using the model Brazilian tensile strength and UCS tests. The mineralogy and grain size models were modeled by directly prescribing the desired properties. In order to simulate the different fabric types, representative thin sections were selected for each fabric type, and then by trial and error the correct mineralogy, grain size, mineral alignment and grain relationships were simulated. Three tests for each sample type and each strength test type were run to obtain a robust test result dataset.
Parametric analysis of the impact of changing geological characteristics was undertaken using the UCS and Brazilian tensile strength test models. The Brazilian tensile strength test models were used primarily to determine the impact of changing characteristics on the strength values, the behaviour of the tensile fracture through the tested simulated sample and to compare with UCS as a measure of brittleness. The UCS test models were also used for the same purpose as the Brazilian tensile strength test models. In addition, the UCS samples were evaluated for their spalling sensitivity based on the behaviour of the induced fractures during sample yield.

Fracture propagation was found to be highly dependent on the sample texture arising from the impact of different mineral-specific strengths, the impact of grain size and grain boundaries and alignment of minerals, especially micas. The modeling demonstrated the ability of mica grains to initiate fracture in stronger minerals while showing that mica is weak along its grain-parallel cleavage. The fractures were found to propagate through micas and feldspars, both of which are aligned and define fabric. In many situations the resultant UCS values were vastly different owing to the impacts from the different textures, be it mineralogy, grain size or fabric. All tested samples showed variations in failure behaviour and strength test values.

**7.1.4 Effect of Spall Sensitivity on Rock Cutting and Tunnel Face Stability**

Numerical modelling of the cutting process was undertaken to investigate the impact of different geomechanical characteristics, in simulated rocks, on the chipping process. These tests were related to the chipping performance analysis presented in Chapter 4, and the spalling sensitivity testing undertaken in Chapter 5, to obtain a relationship between spalling sensitivity and chipping performance. The impact of fabric orientation and stress condition were also investigated for their impact on chipping performance and face stability. The chip area measured from FLAC strain output images was used as the main indicator of chipping performance for analysis of numerical model results.

The texture simulation algorithm described in Section 5 was used to conduct a parametric analysis of chipping performance using the two-cutter model similar to the one used for strength test modelling. The impact of fabric orientation and in-situ stress condition with respect to the tunnel face were investigated using an isotropic stress condition, a biaxial stress condition, and at different biaxial ratios.

The F-Factors were assigned values and used to calculate the Chipping Resistance Factor, which was compared to chip area to calibrate the F-Factor values. The Chipping Resistance

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Factor was applied to TBM performance data from the Southern Aar granite to verify that the system could predict poor chipping.

The impact of face instability on TBM performance was demonstrated for the Southern Aar granite in Chapter 3. The net advance rate and drillability index were used in Chapter 3 to identify locations in the Southern Aar granite where face instability occurred. The same methodology was used to identify the locations with face instability in the Altkristallin. The two-cutter numerical models used to investigate the chipping process, were also used to investigate the preconditioning arising from in-situ stress, for samples with different fabric orientation and type.

The effects of stress condition and fabric alignment on tunnel face preconditioning were examined by modifying fabric orientation to the tunnel face and in-situ stress condition. The analysis was undertaken by running the two cutter models without applying the cutters to the rock surface until the system equilibrated, examining the resulting element failure and plastic strain, then applying the cutters. Although none of the models resulted in what can be interpreted as face instability, they do demonstrate how high stresses will precondition the rock by not only failing elements deep in the rock block, but also by causing considerable strain on a similar order of magnitudes as the strain induced during application of the cutters. With the model data, it is possible to determine Stress-Related Face Stability potential by comparing preconditioning magnitudes at different fabric orientations and under different stress conditions. The different fabric types and intensities were also modeled to investigate the magnitude of preconditioning associated with each fabric type and intensity. These were used to determine risk of face instability leading to reduction of Net Advance Rate arising from different fabric types and intensities.

7.2 Conclusions

7.2.1 Geological Domain Classification

The classified geological domains along the 400m of detailed sample collection in the Southern Aar granite were compared to point load index strength values. The geological domain values ordered with decreasing chipping performance show a clear positive relationship with PLT index strength. The TBM performance domains over the same interval show a negative relationship with increasing domain value and PLT index strength. This exercise shows that geological characteristics can be classified into meaningful domains that correlate well with
laboratory strength, although the relationship between the domain values ordered according to face instability is more complex, likely due to the complex process involved in inducing face instability that cannot be solely characterized using laboratory strength testing. This further supports the use of PLT index strength to characterize samples for excavation design, especially when used in conjunction with geological characterisation. The geological domains were shown to provide meaningful characterisation of the geological characteristics. They were also shown to differentiate between chipping performance categories and tunnel face stability.

7.2.2 Geomechanical Characterisation for Spalling Sensitivity

Based on the TBM performance analysis, numerical calibration and field verification, thresholds and values were associated with the geological factors identified for their impact on the chipping process and face stability. A geomechanical characterisation scheme was defined in which the mineralogy \((F_M)\), grain size and grain size distribution \((F_G)\), and fabric type and intensity \((F_A)\) were weighted according to their impact on the chipping process. Fabric orientation and in-situ stress condition were investigated for their impact on the Stress-Related chipping performance potential \((S_{CP})\). The F-Factors identified for the geomechanical scheme are combined to determine the spalling sensitivity.

The parametric analysis of the impact of different F-Factor combinations on both Brazilian tensile strength and UCS models has provided considerable insight into the failure mode, propagation and ultimate strength values in the following manner:

- The majority of failure in the Brazilian tensile and UCS models is tensile.
- Increased mica content leads to a negative power relationship with both Brazilian tensile strength and UCS because it contains a weakness direction along which slip and strain can occur. Large strains in micas can initiate fractures through adjacent feldspar and quartz grains, while lower stiffness perpendicular to the cleavage planes will inhibit fracture propagation. At low mica content, the mica grains act only as initiators increasing spalling potential, while at very high mica content they generally lower the rock strength, increasing yielding potential. At intermediate mica content the mica grains act as both initiators and inhibitors, decreasing spalling potential without considerably lowering rock strength.
- Increased quartz to feldspar ratio leads to a positive relationship with Brazilian tensile strength and UCS, even though feldspar is slightly stronger in shear and
• Simple increase in grain size facilitates tensile fracture coalescence and propagation and reduces both Brazilian tensile strength and UCS.
• Grain boundaries can act as fracture arrestors around feldspar grains.
• Scale dependence of large-grain samples results in Brazilian tensile strength approaching the highest percentage mineral tensile strength with increasing grain size, or negates the effect of increased grain size in UCS.
• Tensile fracture propagation in Brazilian tensile testing is easiest through aligned and continuous mica grains, more difficult when they must fracture through feldspar bridges and most difficult when they must fracture almost entirely through feldspar and quartz grains.
• The alignment and continuity of mica grains facilitate tensile fracture propagation in Brazilian tensile samples aligned parallel to the loading axis.
• Rapid fracture coalescence and localisation of strain decreases Brazilian tensile strength and is negatively related to mica content in Brazilian tensile samples with fabric oriented perpendicular to the loading axis.
• Increased mica content, alignment and connectivity, as well as mica grain size negatively impact the UCS.
• Shear fracture generation is more difficult than tensile fracture propagation in moderately aligned fabric, resulting in slightly higher UCS.
• The comparison of geological factor changes with two brittleness indices suggests that an index based on the product of UCS and Brazilian tensile strength results in a pronounced relationship. This index could be successfully used to estimate the brittleness of the material in question.
• Point load index strength tests and Brazilian tensile strength tests should be conducted parallel to the tunnel face. In situations where fabric is perpendicular to the tunnel face, these tests should be performed perpendicular to fabric.

The ROC curves for spalling sensitivity during modelled UCS testing demonstrate that grain size is the leading indicator for spalling sensitivity, followed by quartz to feldspar ratio and mica content. In these tests, fabric type and intensity, which were quantified using an $F_A$ weighted for spalling sensitivity, can be used successfully in combination as indicators for spalling sensitivity during modelled UCS testing.
7.2.3 Geomechanical Characterisation for TBM Cutting Performance

The texture simulation algorithm was used to conduct a parametric analysis of chipping performance using the two-cutter model. The mineralogy was investigated in terms of mica content and quartz to feldspar ratio with the following results:

- The chip area, length and depth show a positive relationship with mica content for samples with low (<0.3) quartz to feldspar ratio. At higher quartz to feldspar ratios, this relationship is reversed.
- Rocks with mica content less than 5% and greater than 12% have higher chip areas, length and depths.
- Very low and high mica content have positive impacts on chipping, whereas there exists a mica content between 4% and 12% where chipping performance in reduced.
- Chip area decreases with increased quartz to feldspar ratio.
- The samples with mica content around 12% have a weak decrease in chip area, with values typically lower than for samples with lower mica content.
- Chip area increases with increasing grain size.
- Only one mineral preferred orientation (MPO) sample was tested because TBM data analysis showed that narrowly spaced MPO did not greatly impact chipping performance, and was not found to promote chip formation.
- Chip area was larger for samples with schistosity rather than cleavage.
- Chip area decreases with increasing schistosity intensity.
- Chip area increases with increasing cleavage intensity.
- Fabric oriented 60° and 30° to the face lead to the highest chip area.
- The major stress orientation with respect to the face will have a greater impact on tunnels in which the fabric is oriented parallel to the tunnel face.
- The samples subjected to isotropic stress showed decreasing chip area with increasing stress condition and increased isotropic stress will lead to thinner chips and thus decreased chipping performance.
The samples subjected to biaxial stress show generally increasing chip area with increasing biaxial stress and for samples with fabric oriented oblique to the tunnel face increasing biaxial stress will lead to larger chip formation.

The sample subjected to biaxial stress at different ratios show different results depending on fabric orientation. The chip area decreases with stress ratio for samples with fabric parallel to the tunnel face, while they increase in area for samples with fabric oblique to the face.

The spalling sensitivity, when combined with laboratory strength data and Stress-Related Chipping Potential (Figure 7.1), can be used to determine Chipping Resistance Factor ($C_R$) calculated according to:

$$C_R = F_M \times F_G \times F_A \times \text{labstrength} \times S_{CP}$$  \hspace{1cm} (7.1)

Thresholds were identified for the Chipping Resistance Factor based on laboratory strength values from UCS, UCS and Brazilian, or point load testing beyond which the risk of poor chipping conditions is elevated (Table 7.1). The relationship between Chipping Resistance Factor $C_R$ and chip area cannot yet be defined by a formula with which specific production values could be obtained, however, preconditioning thresholds have been defined beyond to determine the risk of poor chipping (Figure 7.2 and 7.3). The goal of this investigation is to improve prediction of problematic excavation conditions, of which poor chipping is one. Obtaining a value for $C_R$ makes it possible to assess the risk of the excavation conditions leading to poor excavation.
Figure 7.1: Flowchart demonstrating input data for calculating Chipping Resistance Factor $C_R$.

Table 7.1: Summary of thresholds for Chipping Resistance Factors

<table>
<thead>
<tr>
<th>Chipping Resistance Factor Type</th>
<th>Low Risk of Poor Chipping</th>
<th>Moderate Risk of Poor Chipping</th>
<th>High Risk of Poor Chipping</th>
</tr>
</thead>
<tbody>
<tr>
<td>$F_{SSA} = F_A \times F_G \times F_M \times S_{CP}$</td>
<td>$&lt;1.3$</td>
<td>$1.3-1.5$</td>
<td>$&gt;1.5$</td>
</tr>
<tr>
<td>$C_R = F_{SSA} \times UCS \times BTS$</td>
<td>$&lt;1500$</td>
<td>$1500-4000$</td>
<td>$&gt;4000$</td>
</tr>
<tr>
<td>$C_R = 750 \times F_{SSA} \times PLT_d$</td>
<td>$&lt;1500$</td>
<td>$1500-4000$</td>
<td>$&gt;4000$</td>
</tr>
</tbody>
</table>

Figure 7.2: Schematic sigmoidal function relating Chipping Resistance Factor and TBM performance
7.2.4 Geomechanical Characterisation for Tunnel Face Stability

The fabric type and intensity, orientation and in-situ stress condition were used to determine the risk of reduced net advance rate arising from fabric orientation and stress state combinations, as well as fabric type and intensity. The magnitude of preconditioning allows identification of risk of NAR reduction in a qualitative fashion, based on rock preconditioning, arising from fabric orientation and in-situ stress condition. The FA factor used in chipping performance analysis is qualified for its impact on risk of net advance rate reduction, arising from face instability due to preconditioning. These factors can be used in combination to determine the risk of face instability and risk of reduction of net advance rate arising from preconditioning given the availability of fabric and in-situ stress data. This can be used in combination with \( C_R \) to anticipate TBM performance in terms of chipping and face instability (Figure 7.4).
Figure 7.4: Schematic NAR-DI graph showing impact of Chipping Resistance Factor ($C_R$) and preconditioning on TBM excavation performance.

### 7.3 Recommendations

Point load index strength data were successfully used in the calculation of the Chipping Performance Factor in the Southern Aar granite. These PLT-based thresholds hold true for the in-situ stress conditions encountered during excavation of the SAG, but it is not possible to say if these thresholds can be extrapolated to other conditions. Point load testing should be further investigated for its ability to predict chipping performance under other excavation conditions.

The domain classifications were found to relate to geological characteristics, showing that they can capture changes in geology. These analyses show that a simple classification of rock domains at the 100’s of metre scale using field mapping, core logging, or information from previous tunnelling can provide a method by which domains at high risk for poor chipping and/or face instability can be identified and the extent of these domains can be delineated to determine if their impact on chipping performance and face stability will be critical to the project. This should be undertaken early in the excavation design stage so that high risk for poor chipping or face
instability domains are identified and more detailed investigation undertaken to determine the nature of the impact the geology will have on chipping performance and face stability.

The mineral-specific constitutive models were developed using sparse mineral-specific strength data and circumstantial evidence from published rock yielding examinations in which mineral-specific behaviours were described. The constitutive models were verified and calibrated by mineral-specific testing.

The numerical modelling undertaken shows how realistic rock analogues can be used to investigate complex yielding behaviour and perform parametric analyses in which the parameters are explicitly varied. This is a promising approach and with further verification and calibration of the minerals that make up rocks, this approach could be used for a variety of applications. In particular, the constitutive model must be calibrated to be able to model rock behaviour at all ranges of confinement. Other mineral-specific features, such as microfractures, should be included in the model, as opposed to the methodology of decreasing tensile strength for microfractured mineral types, for a more rigorous investigation of the impact of microfractures on rock yielding processes.

The numerical modelling was undertaken in two dimensions for reasons of simplicity and computation time, but three-dimensional models of Brazilian, UCS and two-cutter geometries would be more rigorous and may provide better results. In any case, the comparison of two- and three-dimensional modelling of real rock analogues would be very interesting.

The Chipping Resistance Factor and magnitude of preconditioning were developed and tested on different rocks with similar basic mineralogy: mica, quartz and feldspar. Calibration of the Geomechanical Characterisation Scheme should be undertaken for other rock types, especially those with mafic minerals. Quantification of the impact of preconditioning should also be undertaken with rigorous fabric orientation and in-situ stress condition data collection. In addition, conditions under which face instability is encountered in isotropic rocks should also be used in quantification of factors leading to face instability.

The examination of indicators of brittleness and spall sensitivity based on fracture initiation, propagation and coalescence data from UCS tests, Brazilian tensile strength tests and point load index strength tests should be rigorously investigated. The relationship between Geomechanical Characterisation and brittleness and strength-based spalling sensitivity could then be thoroughly investigated, perhaps identifying meaningful laboratory-based strength tests that relate to rock yielding processes at excavation boundaries. Eventually this approach could be used to correlate geomechanical characteristics with the damage initiation threshold and the interaction of in-situ stress condition and heterogeneity with the spalling limit.
References

ABAQUS Inc. Finite Element Analysis.


Systat Software, Inc. 2006. Sigmaplot


Weh, M. 2003. Personal communication.


Appendix A

This appendix contains documents related directly to the material presented in Chapter 1.
Appendix A.1 : Canada-US Rock Mechanics Symposium Manuscript

The following document was submitted for publishing by the Canada-US Rock Mechanics Symposium. The manuscript was reviewed by C. Frenzel and all changes suggested by him were undertaken. It was also reviewed by the conference committee and changes were made to the manuscript to address the suggestions relating to improved clarity of the goal, application and success of geomechanical characterisation for TBM tunnelling.
Geomechanical characterisation of massive rock for deep TBM tunnelling

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ABSTRACT: A combined geological and rock mechanics approach to rock behaviour prediction, based on improved understanding of brittle fracture processes during TBM excavation, was developed to complement empirical design and performance prediction for TBM tunnelling applications in novel geological conditions. A major challenge of this research is combining geological and engineering languages, methods and objectives to construct a unified geomechanics characterisation system. The goal of this system is to describe the spalling sensitivity of hard, massive, highly stressed crystalline rock, commonly deformed by tectonic processes. Geological, lab strength testing and TBM machine data were used to demonstrate the impact of interrelated geological factors, such as mineralogy, grain size, fabric and the heterogeneity of all these factors at micro and macro scale, on spalling sensitivity and to combine these factors within a TBM advance framework. This was achieved by incorporating aspects of geology, tectonics, mineralogy, materials strength theory, fracture process theory and induced stresses.

A.1.1 INTRODUCTION

Tunnel Boring Machines (TBM) are being used for tunnel excavation in deeper and tougher environments than ever before. Current TBM design and performance prediction methods based on empirical databases and lab scale and in-situ testing (Bruland 1998, Rostami et al. 2002, Rostami et al. 1996, Zhang et al. 2003) consider factors such as UCS, Brazilian tensile strength, joint characteristics (Barton 2000, Sapigni et al. 2002) and TBM specific index test values (CERCHAR 1986, Dollinger et al. 1999, Plinninger et al. 2003). These design methodologies can
be successfully applied to projects that fit within the realm for which the empirical databases contain a large amount of data. When attempting to apply these methodologies to novel conditions, such as in very hard, massive or foliated, unjointed rock at high stress, the nuances that are important under these conditions may be overlooked during TBM design.

Advances made to TBM performance prediction in massive rock consider rock mass fabric (Büchi 1988), but do not provide the necessary distinction between individual rock types and expected behaviour. A geomechanical characterisation system based on an investigation of the impacts individual geological characteristics have on rock fracture and TBM performance will allow engineers to more specifically predict rock response at the face to tunnel and stress geometry during TBM excavation.

Techniques used by geologists to characterise massive rocks, such as rock type classification, are applied to engineering problems to provide greater insight into the differences that are not revealed by conventional engineering methods, such as standard lab strength testing. The purpose of this paper is to present new approaches that quantify geological characteristics based on rock mechanics principles to improve rock behaviour prediction at the tunnel face.

A.1.2 DEVELOPMENT OF A GEOMECHANICAL CHARACTERISATION SYSTEM

A review of recent deep TBM tunnelling projects in the Swiss Alps has shown that, except for in a few extreme cases (Bonzanigo & Opizzi 2005, Burkhard & Isler 2005), the geological prediction was similar to the rock actually encountered during tunnelling (Chopin 2005, Frei & Breitenmoser 2005). The selection of appropriate excavation and support tools and techniques, therefore, critically depends on adequate prediction of rock mass behavior in response to tunnelling for each geological domain within the geometrical and mechanical tunnelling framework (Kaiser 2005). The need for quantifying geological descriptions for engineering geology applications such as open pit mine wall stability (Hoek 1999) and deep, hard rock, tunnel stability (Kaiser 2005) has been demonstrated but to effectively accomplish this, the rock behaviour and response must first be understood in order to define the values of importance for quantification (Kaiser 2005). In-situ behaviour can vastly differ from laboratory behaviour, depending heavily on textural properties (Diederichs et al. 2004), making understanding rock behaviour at the excavation boundary critical to properly quantifying geological characteristics.

The development of the characterisation scheme followed the procedure outlined in Figure A.1.1. The goal for this characterisation scheme was the development of a tool by which geological characteristics could be translated into indicators of susceptibility to spalling failure at
the TBM cutter (small) and tunnel face (large) scales (Fig. A.1.2). The ranges of rock types considered are massive rocks, with Rock Mass Rating (RMR) greater than 75 and Geological Strength Index (GSI) greater than 70. The characterisation scheme focuses on rock behaviour and response that leads to spalling-type yield, where sudden failure is induced through intact rock at the excavation boundary (Fig. A.1.2).

Existing methods, such as RMR (Bieniawski 1989), and Q (Barton et al. 1974), were developed to address conventional blocky ground issues of support and excavation in zones of raveling of the wall and face due to jointed rock masses. Several methods for TBM design, such as Q_{TBM} (Barton 2000) and the NTNU method (Bruland 1998), and TBM analysis methods developed by Büchi (1998) incorporate rock mass characteristics.

Non spalling-type failure through massive rock, resulting from shear failure, does not occur by the same failure mechanism that leads to spalling-type failure. For this type of failure, conventional rock characterization, such as UCS, Cerchar Index (Dollinger et al. 1999) etc, have been shown to be sufficient for TBM design methods such as the CSM method (Bruland 1998, Rostami et al. 1996).

Geological tools employed in this research include petrographical and textural rock description at the thin section, hand sample, and tunnel scale. Engineering tools include point load testing of drill core, rock mass classification (RMR and GSI), and TBM performance (penetration rate and thrust magnitude) data analysis. Materials science principles relating crystal deformation, and stiffness and strength properties (Illston et al. 1979, Nicolas & Poirier 1976) provided direction for interpreting the impact of tectonic deformation on rock strength.
A.1.3 A GEOMECHANICAL CHARACTERISATION SCHEME FOR MASSIVE, HARD ROCK

The geomechanical characterisation scheme was constructed to translate information available through geological description into information that relates directly to rock behaviour, focusing on spalling sensitivity as it impacts chipping and face instability. The flowchart in Figure A.1.3 shows how geological sample characterisation factors are combined to obtain estimates of spalling sensitivity and fracture potential, which are used to make interpretations about the behaviour of the rock at the excavation boundary, either at the cutter or the tunnel face scale. A description of the development of the characterisation scheme summarized in Table A.1.1 is presented in Villeneuve et al. (in prep).
Figure A.1.2. Schematic diagram illustrating the failure mode areas of focus. Top: schematic penetration rate vs gross thrust graph shows two separate processes during TBM cutter excavation: grinding at low thrust and penetration rate, versus chipping (a process akin to spalling) at high thrust and penetration rate. Bottom: tunnel cross sections demonstrate wall and face failure mechanisms; from top left: blocky ground resulting from discontinuities, squeezing due to shearing in low competence rock masses with respect to induced stress, spalling in the wall and/or face, depending on rock mass characteristics and induced stress geometry, and stress-fabric interaction inducing block formation and instability in the face.

All designations for low, medium and high, denoting relative impact on fracturing and spalling behaviour, are related to the cutter-rock interaction and face instability realm. Low impact indicates characteristics that are unfavourable to spalling, and high impact indicates characteristics that promote spalling. This characterisation only suggests sensitivity to spalling and fracture potential, since the manifestation of spalling during TBM excavation also depends on the interaction of the tunnel, anisotropy and induced stress geometries specific to each tunnelling situation.

A.1.4 RELATIONSHIP BETWEEN THE CHARACTERISATION SCHEME AND GEOLOGICAL DESCRIPTION/HISTORY

Testing and calibration of the geomechanical characterization is demonstrated by combining available geological, engineering and mechanical data along a length of tunnel and then dividing them into domains. The rock characteristics, behaviour and response to TBM tunnelling are compared and contrasted within and between domains.
Figure A.1.3. Characterisation schematic showing data collection, classification and combination to obtain fracture potential. Legend: FMM – mineralogy major; FMA – accessory minor; FM – mineralogy; FGP – grain size petrological; FGT – grain size tectonic; FGD – grain size distribution; F – grain size and grain size distribution; FAF – fabric type; FAD – fabric scale; F – anisotropy; FSS – spalling sensitivity; FSSA – spalling sensitivity with anisotropy; FFI – isotropic fracture potential; FFA – anisotropic fracture potential

Table A.1.1. Description of geological characterisation factors

<table>
<thead>
<tr>
<th>Factor</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>FM</td>
<td>Mineralogy. Total and relative percentage of major minerals, FMM, such as quartz, olivine, feldspar, calcite, amphibole, and pyroxene, and total and relative percentage of accessory minerals, FMA, such as biotite, muscovite, garnet, pyrite and magnetite, are weighted for their low, medium or high impact on fracturing and spalling behaviour. The combination of the two results in a low, medium or high designation for the mineralogy factor.</td>
</tr>
<tr>
<td>FG</td>
<td>Grain size and grain size distribution. Median grain size, FGP, grain size reduction due to tectonic processes, such as subgrain formation and grain boundary migration, FGT, and grain size distribution, primary or secondary resulting from tectonic deformation, FGD, are designated low, medium or high, and are combined to result in a low, medium or high designation for grain size and grain size distribution impact on fracturing and spalling behaviour.</td>
</tr>
<tr>
<td>FA</td>
<td>Anisotropy. Foliation type, FAF, and foliation dimension, FAD, in combination as FA, are assigned a low, medium or high designation for impact on fracturing and spalling behaviour.</td>
</tr>
<tr>
<td>FSS</td>
<td>Isotropic spall Sensitivity. F and FG are combined to determine the low*, medium or high sensitivity to isotropic spalling.</td>
</tr>
<tr>
<td>FSSA</td>
<td>Anisotropic spall sensitivity. FSS and FA are combined to determine the low*, medium or high sensitivity to anisotropic spalling.</td>
</tr>
<tr>
<td>FFI and FFA</td>
<td>Isotropic and anisotropic fracture potential, respectively. Standard lab strength values and FSS, for isotropic rocks, or FSSA, for anisotropic rocks, are combined to determine the fracture potential of the rock, and normally manifests itself as a reduction of the lab strength value, representing excavation boundary strength: ( F = F \times \text{lab strength} )</td>
</tr>
</tbody>
</table>

* ‘Low’ designations approach 1, while ‘high’ designations approach 0.45

A.1.4.1 Geological Background

The characterisation scheme is used to quantify the geology of a section of the Gotthard Base Tunnel (GBT) in the central Swiss Alps (Fig. A.1.4). Samples were collected from the Southern Aar Granite (SAG), a unit of the Aar Massif located at the North end of the tunnel (Fig. A.1.5). The Aar Massif is composed of a northern and southern gneiss zone of poly-metamorphosed basement rock, which underwent the highest grade of metamorphism, a core of Central Aar...
Granite (including the SAG), which underwent lower grade metamorphism, and metavolcanics. Alpine foliation dips and becomes steeper and more intense to the south (Trümpy 1980). Alpine metamorphism increases southwards (Masson 1980) to greenschist facies (Keller 1999).

The SAG is mostly composed of granitic gneiss with quartz, feldspar and low to moderate (5-25%) mica content (Keller 1999) making up the major mineralogical components. The fabric varies from granitoid to intensely foliated schist on a scale from one metre to several tens of metres.

A.1.4.2 Geomechanical characterisation of a Subsection of the Southern Aar Granite

The 100 metre long tunnel section being characterised is located at the southern boundary of the SAG and is composed of unfoliated granite with abundant quartz and low mica content, as well as moderately to highly foliated schist with nearly equal quartz and feldspar content, and medium mica content. This section was selected due to its geological and mechanical variability, as well as the availability of 5 cm diameter drill core samples at one metre spacing. Figure A.1.6 is a compilation of mapping of the tunnel wall, division into geological domains, point load (PLT) index strength data and the values resulting from the characterisation of the thin sections.

On the tunnel-wall maps, macro geological features such as shears and fractures, as well as distinct changes in rock type were recorded, supplemented by locations of overbreak and spalling. Of particular interest are sections of higher quartz content with corresponding lack of fabric, which manifest themselves as several metre wide zones or centimeter wide veins. Using the drill core samples and tunnel wall map records the geology along the section was classified into the dimensionless domains, outlined in Table A.1.2, using the following criteria: mineralogical components, median grain size, grain size distribution, fabric type and overall variability, in terms of shear zones, fractures, rock type and tunnel wall overbreak.

Figure A.1.4. Map of Switzerland showing Gotthard tunnel location outlined in dashed box, Alps shown in shaded areas. Modified from Schweizerischen Geologischen Kommission.
Figure A.1.5. Geological cross section through the major units of the Gotthard Base Tunnel (after Keller 1999). Southern Aar Granite shaded at southern end of the Aar Massif. Vertical scale same as horizontal scale. Representative hand sample photos (on left) and photomicrographs (on right) associated with three locations along the Gotthard Base Tunnel transect.
Figure A.1.6. Cross-sectional compilation of tunnel wall mapping data, rock mass domain classification, point load index strength (2m central average), $F_M$, $F_G$, $F_{SS}$, $F_A$, and $F_{SSA}$ factors, TBM performance (NAR and DI values, on a per stroke (2m) frequency) and overbreak record data.
Table A.1.2. Description of geological domains from Figure A.1.6

<table>
<thead>
<tr>
<th>Domain</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>&gt;10 metre scale variability with feldspar, quartz and mica, in decreasing percentage, medium (0.5-5mm) grain size and fabric ranging from preferred orientation of feldspars to schistosity defined by micas, no spalling</td>
</tr>
<tr>
<td>B</td>
<td>&gt;10 metre scale variability with feldspar, quartz and mica, in decreasing percentage, medium (0.5-5mm) grain size and fabric consisting of schistosity defined by micas, approximately 5% spalling</td>
</tr>
<tr>
<td>C</td>
<td>&gt;10 metre scale variability with feldspar, quartz and mica (up to 30%), in decreasing percentage, small (&lt;0.5mm) grain size and fabric consisting of continuous cleavage defined by micas, nearly 30% of area contains spalling (spalling not shown in Figure A.1.6)</td>
</tr>
<tr>
<td>E</td>
<td>Less than decameter scale variability with feldspar, quartz and mica (high variability from 2-25%), in decreasing percentage, medium (0.5-5mm) grain size, with micas often &lt;0.5mm and feldspars often &gt;5mm, and fabric ranging from preferred orientation of feldspars to schistosity defined by micas to cleavage defined by micas in narrow shear zones, 10-25% of mapped area contains spalling (spalling not shown in Figure A.1.6)</td>
</tr>
<tr>
<td>G</td>
<td>&gt;10 metre scale variability with feldspar and quartz (mica only ~2%), medium (0.5-5mm) grain size and no fabric, no spalling and infrequent shear zones</td>
</tr>
</tbody>
</table>

* note that domains D and F are not shown in Figure A.1.6

Thin sections were cut from the drill core samples, which had previously been strength tested with a point load testing apparatus, for which results are shown in Figure A.1.6. The thin sectioned samples were characterised according to the scheme shown in Figure A.1.3. The values assigned to \( FM \), \( FGs \), and \( FA \), and their combinations, \( FSS \) and \( FSSA \), are shown in separate graphs in Figure A.1.6.

A.1.4.3 Geomechanical Characterisation and its Relation to Encountered Geology

The rock mass domain classification correlates well to the geology recorded in the tunnel wall map in Figure A.1.6. Some mismatch exists due to the heterogeneity of the rock types, the orientation of their boundaries in relation to the orientation of the drill core (roughly 6 degrees to horizontal) and the resolution of the drill core diameter. Domain G corresponds to zones of quartz-rich rock in the tunnel wall map, domain E corresponds to low shear zone spacing, domain A corresponds to rare shear zone occurrences and domain B corresponds well to sporadic shear zone occurrences.

A few trends in the characterisation factors can be observed. The higher the \( FSS \) value, the lower the spall-behaviour potential of the rock, and vice versa. The higher \( FSS \) values correspond to domains A and G, with one high \( FSS \) in domain E (14m). The medium to low \( FSS \) values correspond to domains A, B and E. One occurrence of low \( FSS \) corresponds to domain G (63m), but this is a very coarse grained sample of limited extent.

Similarly, the higher the \( FSSA \) factor, with the impact of foliation taken into consideration, the lower the spalling potential. A large number of the samples in the tunnel area in Figure A.1.6 correspond to domain G or are samples that have no or poorly developed foliation, and \( FSSA \) is the
same as \( F_{SS} \), but the samples that do have foliation and have a decreased \( F_{SSA} \) correspond to domains A, B and E, which are generally characterised as having well developed foliation.

A comparison of the domains and the PLT index strength shows a correlation between degree of variability and the domain: highly variable domain E also exhibits high variability in the PLT, the same is true to a lesser extent for domain B, the variability is low for low variability domain A, while the PLT is homogeneously higher for domain G.

\[ A.1.4.4 \text{ Geomechanical Characterisation and its Relation to Geological History} \]

The domain and geological factor characterisation of the focus area in Figure A.1.6 demonstrates both homogeneity and heterogeneity in the rock unit. Most of the domains are selected based on broad homogeneity of characteristics such as mineralogy, texture and macro features, while domain E is selected based on its heterogeneity. In general, however, all of the domains have similar mineralogy (mainly composed of varying percentages of feldspar, quartz and mica, in decreasing order) and grain size (small to medium). The foliation varies slightly between alignment of feldspars, schistosity defined by mica, and decimeter scale shear zones with continuous cleavage (non-parted), with domain G lacking foliation.

The geological description from section 4.1 is very similar to the domain descriptions in this zone. The domains capture the geological descriptions written by geologists for tender purposes, while at the same time distinguishing boundaries between zones with slightly different characteristics. In addition, the domains and geological factor characterisation highlight the different geological characteristics that impact the rock behaviour during TBM excavation.

\[ A.1.5 \text{ RELATION OF GEOMECHANICAL CHARACTERISATION TO ENGINEERING PROCESSES} \]

\[ A.1.5.1 \text{ Examination of TBM performance} \]

Engineering data were collected from the geological section described in section 4. TBM performance data were collected from the TBM used to excavate the northern section of the GBT, at a depth of approximately 2000m. Figure A.1.7 shows the relationship between the penetration rate (mm/revolution), the gross machine thrust (kN) and the net advance rate (single TBM stroke distance normalized by active tunnelling time, during which the head was turning, in mm/min). The net advance rate can reflect decreases to advance resulting from either low penetrability or
face instability. In addition to TBM data, location and magnitude of overbreak at the tunnel wall and records of tunnel face overbreak are also shown in Figure A.1.6.

### A.1.5.2 Application of Geomechanical Characterisation Scheme to TBM Excavation

Keller (1999) states that it is difficult to determine where spalling is likely to happen in the tunnel section. Domains for macro characteristics and geological factors for micro characteristics can be used to interpret the rock behaviour during TBM excavation. By looking only at the PLT data in Figure A.1.6, zones with negatively impacted advance rate should correspond to low PLT index strength, but this is untrue around 15m, while a comparison of PLT data and overbreak data shows a good correspondence between index strength and magnitude of overbreak.

When PLT and $F_{SSA}$ are observed simultaneously (as an approximation for generating $F_{FG}$) and compared to the TBM relationships and overbreak records, a pattern emerges in which combinations of low PLT index strength but high $F_{SSA}$, such as at 15m, or combinations of low PLT index strength and low $F_{SSA}$, such as at 30-35m, can result in minor overbreak in the tunnel face that does not negatively impact the rock toughness, while combinations of high PLT index strength and high $F_{SSA}$, such as at 53-58m, can lead to stable face conditions but sudden delayed wall overbreak once the face has progressed some distance away from this area. Combinations of moderate PLT index strength and low to moderate $F_{SSA}$, such as at 77-81m, can lead to face instability with negative impact on TBM advance.

When analyzing this data, an understanding of the spatial extent of rock types is necessary to interpret the impact on rock behaviour during TBM excavation. For example, a low $F_{SS}$ resulting from large grain size in the unfoliated rock from domain G (63m) would suggest high spall potential, but no evidence of face or tunnel wall instability exists. The extent of this grain size extreme is less than 2 metres and did not impact the behaviour enough to affect the TBM performance relationship in this area. A rock domain with high variability at TBM scale (such as domain B), in particular in terms of spall sensitivity and strength, will lead to higher magnitude of face instability and neutral to negative impact on TBM advance, while a rock with low heterogeneity in strength and spall sensitivity (such as domain G) will lead to homogeneous impact on TBM advance. In the case of domain G the impact is negative due to unfavourable combinations of high strength and low spall sensitivity, but the contrary is possible with favourable combinations of moderate-high strength and moderate spall sensitivity.
A.1.6 CONCLUSIONS AND FUTURE WORK

When designing the excavation and support methodology for TBM tunnelling it is critical to make appropriate predictions for rock behaviour and response based on the geological data available. The interpretations presented here are used to demonstrate the basis for the development of a geomechanical characterization scheme used to predict spalling sensitivity and fracture potential at the tunnel face. This scheme is a two-fold approach considering micro scale features such as mineralogy and texture as well as macro scale features such as scale of rock type variability and was illustrated with an application to a 100m long tunnel segment excavated by TBM.

The ability of the geomechanical characterization to capture the geological description was demonstrated by comparing it to tunnel wall maps and geological descriptions of the SAG rock unit. Some preliminary relationships between the characterisation and the rock behaviour during excavation were also demonstrated with TBM data and tunnel wall and face overbreak records. Further calibration of the geomechanical characterisation scheme aims to refine the quantitative approach for relating geology to TBM performance.

The discussion surrounding Figure A.1.6 also demonstrates the need for attention when employing the geomechanical characterization methodology during site investigation. Geological descriptions should contain appropriate information to determine different domains based on geological features and degree of variability. Within each domain representative samples should be selected for geological factors characterisation. In addition, if samples exhibiting extreme
characteristics with respect to the domain are selected for geological factors characterisation, then the scale and frequency of the extreme geology should be taken into account when associating it to rock behaviour during TBM excavation.

The quantification and classification of geological characteristics for rock behaviour prediction has been illustrated at the micro to several metres scale. In situations where detailed data is unavailable the geological description may be on the rock unit (100’s of metres) scale. Future work will address the development of a methodology for combining the benefits of domain and geological factor characterisation at various scales, from mm scale in thin sections to 100’s of metres from rock unit descriptions, into a meaningful system applicable at the TBM scale.

A.1.7 ACKNOWLEDGEMENTS

Financial and technical support provided by Herrenknecht AG. Financial support also provided by a Postgraduate Scholarship from the Natural Science and Engineering Research Council (NSERC) of Canada. The authors would like to thank Mr. Scheifele and Mr. Eggel from Herrenknecht Schweiz, Mr. Holzhuber from AGN, Mr. Bertholet and Dr. Weh from MaTrans, and Mr. Classen and Mr. Ghirlanda from TAT for their support during data collection, and Prof. Löw at ETH Zürich for help obtaining testing equipment.

A.1.8 REFERENCES


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Appendix B

This appendix contains documents related directly to the material presented in Chapter 2.
Appendix B.1 : TBM Performance Data Collection and Processing Methodology

B.1.1 Introduction

TBM performance data were obtained by a data acquisition system (DAS) operated by Herrenknecht AG. The details of the DAS are not available and will not be discussed here. The TBM data are formatted into a spreadsheet containing entries for equipment status every 10 seconds. Each individual spreadsheet file represents one stroke of the thrust pistons, roughly 2m advance distance. The majority of the data contained in the files is not used, but the data representing thrust, torque, RPM, speed, penetration rate and the surveyed location of the head based on the extension of the thrust pistons comprises the data used throughout this research.

B.1.2 Start-up Testing

A method by which TBM performance is related to rock toughness in stable face conditions was developed. The TBM thrust is increased by very small increments from full stop to maximum thrust. This results in high resolution TBM data for this interval and is necessary because data is collected every 10 seconds and under normal operating conditions only 2-5 points are collected in this low-thrust region, while by undertaking this methodology 30-50 points can be collected. The data of interest from this test are penetration rate and thrust. The slope of the penetration rate versus thrust graph, and the inflection point at which the slope changes, are of interest to this research and are obtained using the start-up test.

B.1.3 Friction on TBM Head

The thrust value obtained from the data acquisition system (DAS) is gross thrust relating to the amount of force exerted by the thrust pistons. This thrust incorporates the impact of friction on the TBM head, and is not related geological conditions at the tunnel face. It is assumed that there are other contributions to thrust not related to geology but they are difficult to identify and even more difficult to remove from gross thrust. The friction contribution to thrust requirements can be estimated by observing the maximum thrust requirement during advance or reversal of the TBM head in situations where it is not in contact with rock at the face. This occurs frequently, in fact, during nearly every stroke.
In order to estimate this value, a macro was written to find the maximum thrust requirement during TBM free movement. Use was made of the thrust data from the end of each stroke, at which point the machine is pulled back from the face a short distance. The macro searches through data points in which the thrust magnitude is increasing (TBM start-up) but the penetration rate and torque requirements are zero. The highest thrust detected during these criteria is recorded for that stroke. The thrust values during slow-down are not used because it is assumed that speed (and therefore penetration rate) is only calculated when the machine is actively thrusting. Using these values results in erroneously high thrust values for friction because the penetration rate drops to the zero immediately once the machine is stopped but residual thrust force continues to be recorded by the DAS until it reaches zero. The thrust values during start-up are also not used because it is not possible to decipher whether the machine is in contact with the face or not. If it is, then it is not moving freely and some of the thrust will reflect the force exerted by the machine on the face, in addition to friction, which would lead to erroneously high measures for friction.

The thrust requirements related to friction are not used for on a stroke by stroke basis, but rather the average of all values in a rock mass. The reason for this arises from the frequency of data acquisition set at 10 second intervals. The thrust increases in a linear fashion but the DAS only records point data at 10 second intervals. During thrust increase the maximum thrust with zero penetration rate may occur between data acquisition points, thereby underestimating the friction. The data are processed for standard statistics and a histogram of all values. The outlying data are excluded from a second statistical processing, to obtain the most applicable friction data. The median values are used, although they are generally within 1-2% of the average values, because they represent the central value within the dataset, thereby further removing the effect of outliers.

The friction is related to several factors including rock roughness and radial force exerted onto the tunnel walls by the pressure shield. These may vary from one stroke to another, and the best friction estimate would be made on a stroke by stroke basis, but for the reason stated above, an average must be used. The friction estimate can only be considered as an estimate, but it does improve the gross thrust measurement. Once the friction is removed from the gross thrust, a value for thrust per cutter can be obtained, which can then be used to compare machines with different diameters. Only Herrenknecht machines are compared in this research and comparison between TBM manufacturers has not been undertaken, and it is not clear if such an approach would be valid.
Appendix B.2 : Tectonic Deformation Processes

B.2.1 Introduction

This appendix is a primer on deformation and recovery of grains in rock, with comparisons to analogues from metallurgy. Much work has been accomplished in the area of metallurgy regarding material deformation and recovery with the purpose of investigating their effects on the strengths of metals. These principles of material strengthening and weakening by disturbance and recovery can be used to understand the strength of tectonically deformed rock. Evidence of tectonic deformation and recovery in rocks can best be seen by scanning electron microscopy, they can also be found by transmitted light microscopy of rock thin sections.

B.2.2 Defects

There are several types of defects (Ilston et al., 1979):

- Vacancies are missing atoms within a lattice
- Interstitials are extra atoms within a lattice
- Impurities are extra atoms of a different type
- Dislocations are incomplete crystal lattices
- Grain boundaries (B.2.1) and edges are 2-D defects
- Pores and cracks are 3-D defects

Dislocations occur naturally in crystal and can be induced by deformation from: existing dislocations, defects (voids, interstitials, impurities), grain boundaries (Figures B.2.1), and crystal surface irregularities.
Deformation of grains increases the internal energy of the system and randomizes the crystal lattice of the grain, and occurs during dynamic tectonic processes under increased pressure and temperature (Nicolas and Poirier, 1976). The major 2-D defects are dislocations and twins, while 3-D defects are voids within the rock matrix. Defects are formed during straining of a material and the type of defect formation depends on the temperature and strain rate the material experiences (Ilston et al., 1979). Strains can also induce defects to move within the crystal lattice. Movements of defects can lead to strengthening or weakening of a material by creating new defects, concentrating defects in one area of the crystal lattice or annihilating defects to return the crystal lattice to a regular geometry (Nicolas and Poirier, 1976).

Dislocations are line (2-D) defects created when a line in the crystal lattice terminates before it reaches the edge of the crystal. They can be induced to move along the crystal lattice by shear strain, and can glide along the crystal lattice perpendicular (edge) or parallel (screw) to the dislocation line (Ilston et al., 1979).
Dislocations induce a compressive and tensile stress in the adjacent crystal lattice. Two adjacent dislocations can repel each other and block each other’s motion if they have the same orientation. Dislocations can also attract each other and annihilate each other, leading to recovery of a complete crystal lattice, if they have opposite orientations (Nicolas and Poirier, 1976). If dislocations annihilate each other then the crystal has recovered some of its energy and fallen to a lower energy state. If the dislocations repel each other, or pile up at an impasse, they can create a number of types of boundaries. Dislocations have difficulty moving across unfavourably oriented grain boundaries. Pile up of dislocations at an obstacle (void, interstitial, grain boundary) may cause strain hardening. A dislocation wall made up of arrays of edge dislocations leads to sub-grain boundaries (Nicolas and Poirier, 1976).

Twins are 2-D defects formed by mirrored atomic positions across a boundary and frequently form during crystallization (Nicolas and Poirier, 1976). There are three major twin types in plagioclase: albite, Carlsbad and pericline. Albite and Carlsbad twins (Figure B.2.2) are oriented parallel to the grain c axis while pericline twins are oriented parallel to the grain b axis. Carlsbad twins are penetration twins (twins formed when two crystals intergrow) and cut a grain in half and are most frequent in anorthite and orthoclase. Albite and pericline twins are contact twins repeated in a polysynthetic manner: twins form at the planar contact between repeating, parallel crystals. Albite twinning is found in all plagioclase types but pericline twins are more common in anorthite. Albite and pericline twins also occur in potassium feldspar, and especially in microcline and anorthoclase, form together to produce ‘tartan’ twinning (Nesse, 1986).

Twins can also be used in deformation analysis: deformation twins form by crystal deformation and bent growth twins provide evidence of deformation. Deformation twins follow the same formation rules as growth twins but have tapered ends and are always multiple, never simple, and occur in plagioclase and calcite (Nicolas and Poirier, 1976). In plagioclase deformation twins follow the albite or pericline rules and the ends taper towards the centre of the grain (Figure B.2.3). Deformation twinning tends to occur at low temperature and high strain rates (Nicolas and Poirier, 1976). Bent twins (Figure B.2.4) form when a grain is bent under compressive strain.
Figure B.2.2: Photo of thin section of Leventina gneiss showing albite and Carlsbad twins in plagioclase (GB-9C)

Figure B.2.3: Photo of thin section of Leventina gneiss showing mechanical albite twins (GB-9C)
B.2.4 Mechanisms of Recovery

Recovery is a combination of processes that act to reorder the crystals and lowers the strain energy in the system, and are enhanced at higher temperature (Ilston et al., 1979). They can occur during (dynamic) or after (static) deformation. Increased temperature leads to increased movement of dislocations, which are more likely to interact and annihilate each other if positioned favourably (Nicolas and Poirier, 1976). Increased temperature also facilitates dislocation climb around obstacles, relieving pile-ups of similarly oriented dislocations and reducing energy (Ilston et al., 1979).

Increased dislocation movement leads to greater interaction and eventual formation of dislocation walls. The movement and alignment of dislocations is manifested in grains as the progression from undulose extinction, deformation lamellae and finally subgrain boundaries (Figure B.2.5). This process decreases the energy in the grains adjacent to the dislocation walls (Nicolas and Poirier, 1976). Undulose extinction can be seen under crossed polars as a waving extinction passing progressively through a crystal (Figure B.2.6). Subgrains remain part of the crystal but have a lattice orientation differing by ~ 5° and can be seen under crossed polars as having a different extinction angle than adjacent subgrains (Figure B.2.7). If crystals lattices
across subgrain boundary differ in orientation by more than ~ 5° true grain boundaries form and the adjacent crystals can be considered as individual grains (Figure B.2.8) (Nicolas and Poirier, 1976).

Figure B.2.5: Progression of recovery by alignment of dislocations in a crystal (after Passchier & Trouv, 1996)
Figure B.2.6: Photos of thin section of Leventina gneiss showing undulose extinction in a quartz grain (GB-4B)

Figure B.2.7: Photo of thin section of Leventina gneiss showing subgrain boundaries in quartz (GB-2C)
Recrystallisation takes time and increased temperature since it involves diffusion processes. It can be dynamic or static and leads to a grain size reduction and decrease in strain energy in the crystal lattice. Partial recrystallisation is the stage at which a rock has been sufficiently heated to begin the recrystallisation process in some grains, while others remain in the high-energy state. This type of rock displays a bimodal grain size distribution between fully recrystallised grains and those still undergoing recrystallisation (Figure B.2.9).

An accumulation of dislocations can result in a greater than 15° difference in lattice alignment resulting in the creation of new grains from the original grains and leads to grain size reduction (Figure B.2.10). Migration of grain boundaries from low-energy (low dislocation density) grains into high-energy grains leads to irregular grain boundaries (Figure B.2.11). With continued static heating the diffusion continues, resulting in increased grain size and the creation of more regular grain boundaries (Nicolas and Poirier, 1976).
Figure B.2.9: Photo of thin section of Leventina gneiss showing partial recrystallisation of quartz and plagioclase grains (GB-2C)

Figure B.2.10: Photo of thin section of Leventina gneiss showing a quartz grain with subgrain boundaries and a rotated subgrain (GB-4B)
B.2.5 Effect of Deformation and Recovery on Strength

B.2.5.1 Strength of Metals

Damage accumulation and recovery act to increase and decrease the internal energy state of the material. These increases and decreases act to change the ability of the material to deform and make it harder and more brittle or softer and more ductile. Deformation increases material strength by increasing the number of dislocations and the energy state of crystals since: increased numbers of dislocations increases the likelihood of interaction where dislocations block each other’s motion and piled up dislocations will increasingly repel approaching dislocations, increasing the energy required to continue deformation (Ilston et al., 1979). The yield strength of the material will increase during work hardening but the ductility will decrease (Figure B.2.27). Recovery weakens materials by returning the crystals to lower energy states but allows the formation of lower energy grains from higher energy grains and grain growth increases grain size which facilitates dislocation slip, making the material weaker but more ductile (Figure B.2.28).
Figure B.2.12: Three-dimensional graph of effects of stress and percent of cold work applied to a material on the strain it experiences (after ASM 1990)

Figure B.2.13: Effect of recovery of a material by annealing to its tensile strength and ductility (after Sachs and Van Horn 1940)

B.2.5.2 Implications for Tectonically Deformed Rocks
Tectonic processes will increase and decrease the strength of rocks by providing the heat and differential pressure necessary for deformation and recovery of grains. Low temperature – high differential stress tectonic damage of rocks can be considered the equivalent of work hardening of metals, which increases strength by increasing deformation and damage accumulation. High temperature recovery during and after tectonic processes lowers the strength of rocks in the same way that heating and annealing of metals recovers ductility but lowers strength. The geological history of a rock will have an effect on the strength of the rock not only in terms of macro damage, mineralogy and fabric, but also in terms of accumulated and annealed micro-damage that can be observed in microstructures.

Deformation followed by recovery processes will lead to dislocation interactions, manifested as undulose extinction, deformation lamellae, subgrain boundaries and subgrain rotation. Each of these examples of dislocation interaction poses an increasing obstacle to fracture propagation. In addition to the increased plastic deformation strength resulting from the increased concentration of dislocations, there will be an increase in fracture strength resulting from increased grain boundary-type obstacles.

The deformation and recovery processes will also have an impact on grain size and quantity of grain boundaries. Dynamic and static recrystallisation may produce smaller grains with fewer dislocations. The recovery process results in weaker grains due to the reduction of dislocation interactions, but smaller grains have greater grain boundary surface area to interact with fractures, making it more difficult for intragranular fractures to propagate beyond the grain boundary. Both phenomena impact the grain strength in opposing ways, but the impact of grain size reduction on fracture propagation can be directly investigated with the methodologies developed in this research. The impact of mineral grain strengthening and weakening arising from alignment and dispersal of dislocations could not be investigated in this research.
Appendix B.3 : Geological Dataset Collection

B.3.1 Introduction

A data collection methodology was developed during field work in which the most amount of data were collected as possible given the limitations of the work site. A database was developed to contain all digital data, which were input on a regular basis during field work. Paper records of tunnel conditions during tunnelling were also collected and digitally input into the database upon return to Canada. The data were then manipulated and polished for analysis and interpretation.

Rock testing was conducted both onsite and in the rock mechanics testing laboratory in Canada. The data were processed and manipulated and used to characterise the rocks in Chapter 2 as well as in the numerical modelling calibration undertaken in Chapter 4.

B.3.2 Data Collection

Over the course of three visits totalling nearly 12 months, approximately 1500m of tunnel records were collected. The types of data that were collected include:

- Maps of the macro geology visible on the tunnel walls (Figure B.3.1a) immediately behind the TBM head during advance and before shotcrete application;
- Tunnel wall photos, detailed photos of structure and macro photos of fabric (Figure B.3.1b)
- Depth of face failure maps and photos showing face stability conditions (Figure B.3.2)
- Detailed descriptions of geology, structure and rock mass classification (GSI, RMR, Q)
- Hand samples for thin section analysis
- Point load rock strength testing of nearly 450m of tunnel rocks at 1 metre intervals.

In addition to data collected in the tunnel the following were provided by contractors:

- Seismic probe results
- Face maps
- Overbreak records
- Longitudinal geology maps
- 100m interval UCS test results
These were used to supplement the data collected in the tunnel and any testing results obtained from testing of samples in Canada.

These data were used to characterise the rock through which the tunnels were being excavated. Information contained in literature was used to obtain an understanding of the geological history and regional geological setting of the tunnels, but geological descriptions at the TBM stroke scale were made using the maps, photos, samples and strength tests collected and conducted during field work.

Figure B.3.1: Examples of a) tunnel wall maps and b) corresponding close-up fabric photos and c) wall photos
B.3.3 Point Load Testing

B.3.3.1 Sample Selection and Preparation

During the final Amsteg site visit (October-December 2005), exploratory drilling undertaken by AGN presented a unique opportunity to obtain drill core and to undertake an extensive point load testing program. The Engineering Geology Department at ETH Zürich generously loaned their point load testing apparatus for this study. Some background information relating to point load testing procedure and the approach undertaken during this particular testing session are described here.

Point load testing was performed on 54mm diameter diamond drill core taken from the Southern Aar Granite and Southern Gneiss zone rocks. The core was oriented parallel to the tunnel, dipping at 6 degrees into the tunnel floor (Figure B.3.3), which was accounted for as it resulted in a 1m offset in the tunnel metre location over the length of the core when compared to the geology in the tunnel. Samples were collected at the tunnel face to add further control on the location of the core pieces being tested. In this manner, the TBM performance records can be related precisely to the geology, which varies on a 1-3m scale.
The core was drilled by AGN for probing of the ground ahead of the face, therefore, the core was not oriented; pieces were obtained by AGN employees with AGN’s permission. Typically, 10cm long pieces were collected at one metre intervals along three core lengths totalling 385m.

B.3.3.2 Testing Methodology

The point load testing apparatus (Figure B.3.4) was manufactured by Boart Longyear Interfels GmbH in Bad Bentheim, Germany. The number of tests conducted for each interval depended on the lengths of core obtained. Diametral tests (Figures B.3.5 and B.3.6, left) were conducted first and the remaining intact pieces were then used for axial testing (Figures B.3.5 and B.3.6, right).

Figure B.3.3: Schematic diagram showing location of Preventor Boring relative to tunnel axis, and how diametral (face parallel) and axial (face perpendicular) point load tests relate to geometry of tunnel
Figure B.3.4: Point load tester set-up

Figure B.3.5: Schematic of loading directions for diametral and axial point load tests (after Franklin, 1985)

Figure B.3.6: Photo showing loading directions for diametral and axial point load tests
The samples were installed on the testing apparatus and loaded according to ISRM standards \{244\}. Each tested sample was examined for failure plane geometry and only the results from valid tests (Figure B.3.5) were used for PLT strength determination.

### B.3.3.3 Data Processing

The equations for size and shape correction were developed by Franklin \{243 /a\}. The basis for the shape correction is the transformation of the minimum area of failure to its equivalent circular area, and using the equivalent diameter, as follows:

\[ D_e = \sqrt{\frac{4WD}{\pi}} \]  \hspace{1cm} B.3.1

where \( D_e \) is the equivalent diameter, \( W \) is the core diameter and \( D \) is the core height.

\[ I_s = \frac{P}{D_e^2} \]  \hspace{1cm} B.3.2

where \( I_s \) is the point load index corrected for non-circular shape; \( P \) is the pressure in MPa at which the sample failed.

The basis for the size correction is the transformation of the size of the core or sample to an equivalent 50mm diameter core index, as follows:

\[ I_{s50} = I_s \left( \frac{D_e}{50} \right)^{0.45} \]  \hspace{1cm} B.3.3

In most cases the core was oriented perpendicular to the foliation, rendering diametral testing straightforward. Depending on the sizes of the pieces of core it was possible to test it diametrically parallel and perpendicular to the lineation, and those pieces axially perpendicular to the foliation and lineation. Testing in all directions allows the quantification of the anisotropy of the geology; providing the strength in the direction parallel to the face (sensitive to induced stress at the face) and perpendicular to the face (sensitive to stress induced by the cutters).

The anisotropy index (AI) was calculated as suggested in Broch \{242 /a\} as the ratio of the axial and diametral point load index. For cores drilled normal to the foliation:

\[ AI = \frac{I_{s50axial}}{I_{s50diametral}} \]  \hspace{1cm} B.3.4

In strongly anisotropic rock types the difference in strength between these two loading directions is large and may affect the performance of the machine (as will be discussed later). The inverse of the anisotropy index (\(1/AI\)) is used in graphs in this research.
The PLT strength values can be related to UCS values using the ratio 1:24 \cite{Broch241,Franklin241}, but this is an average value for different rock types and can range from 15-50 \cite{Kahraman240}. Picking any of these values can lead to large errors and the only reliable way to relate PLT to UCS is to have both types of test results for the same rock type. The UCS testing done on ten of the samples from the point load test dataset (Section B.3.3.4) was used to calibrate the PLT:UCS strength ratio so that more reliable estimates of UCS strength could be made throughout the dataset. The UCS strength values were compared with the statistics of the UCS estimates (PLT to UCS ratio = 24) based on PLT values within the same rock domain classification (Section 2.2.2). The UCS test strength values were assumed to be similar to the mean of the UCS estimates from PLT tests. The ratio of 24 was used as a first estimate, the statistics of the domain were compiled, and the ratio was adjusted such that the mean was similar to the UCS test values. The basic estimate of 24 \cite{Broch241} was valid for most rock domains, although a ratio of 22 resulted in a better match for domain B.

### B.3.3.4 Data Manipulation

Prior to their use the point load test data at each location were averaged, in addition to the removal of invalid or clearly outlying values. Trendlines with 3-point central averages were then used instead of individual point data when representing data. The 3-point average was selected because it highlights small scale strength variations while reducing the effect of outliers. In the test section, two strength variation patterns are superimposed; a long-wave strength variation roughly at the 200m scale within an $L_{50}$ range of 2 to 8, and a short-wave strength variation roughly at the 10m scale with a variation of about $+1/-1$ to $+4/-1$. The geology was observed to vary at the decimetre to metre scale where units of low, medium and high strength interchanged within short distances. This can be seen in the graphs of rock strength (for example Figure B.3.7) as well as in the graphs of machine performance. The variability is caused by the presence of shear zones, veins and narrow units of more granitic or gneissic rock. This variability due to changes in geology can impact the TBM performance and tunnel behaviour since both potential affect the stability, at the small scale for chipping and at the larger scale for face or wall stability. The resolution of the point load test data is on the metre scale and was used with thin section analysis of the tested samples to relate to TBM performance by combing the two datasets (as described in Section 7.16 Appendix 5.3).
B.3.4 UCS Testing

B.3.4.1 Sample preparation

Ten samples from the exploratory drilling (core geometry described in Section B.3.3.1) were not tested with the PLT apparatus and were shipped to Canada for UCS testing in the Rock Mechanics Laboratory in the Mining Department at Queen’s University under the supervision of Prof. J. Archibald and Dr. P. Dirige. The samples were selected based on length: they were the longest samples available. They were trimmed and smoothed according to ISRM standard {{239}} by Dr. Dirige, with special care taken to produce the longest samples possible, given their originally short length. The length:diameter ratios ranged from 1.9:1 – 2.4:1, which is on the low end of the optimal ISRM range (2:1 – 3:1), but is close enough to produce suitable results for the purpose of this research.
Figure B.3.7: Top: Point load ($I_{50}$) index data in the axial and diametral directions with 3 point floating central average trendline. Bottom: photomicrographs of rock samples corresponding to red points on point load strength.

Coordinate axes were measured onto each sample to locate six microseismic phones for acoustic emission monitoring and one strain gauge to monitor lateral and axial strain (Figure B.3.8). The locations of the phones were measured from the bottom of the sample, and recorded as coordinates, with the bottom centre as 0,0,0. A single strain gauge was used due to limited sample size and for minimal interference with the microseismic equipment. It was secured onto the sample with epoxy glue, ensuring full contact with the rock along the gauge length. The gauge was tested for resistance, leads were attached and soldered and the ends were secured with tape. The microseismic phones were wetted with honey to ensure good conductance with the rock and secured with rubber bands.
The strain gauge leads were connected to an analogue data recorder, which was in turn connected to a computer with software that converted the resistance change to units of microstrain. The microseismic phones were connected to a computer that records both the arrival times of the p-waves and a tally of the microseismic hits recorded by the microseismic phones each second. The microseismic software is also capable of recording information for event tally and event location, but these functions were not used for this research.

**B.3.4.2 Testing Methodology**

Prior to UCS testing the microseismic phones were tested for location and functionality by initiating a seismic pulse from a seventh phone located at the centre top of the sample (coordinates: 0,0,length). If the phones were assessed to be functioning properly, the seventh phone was used to generate several pulses and was then removed. The first arrival times for the pulses were recorded for each microseismic phone and used, with the sample geometry and phone coordinates, to calculate the p-wave velocities of the samples.

For UCS testing the samples were loaded into a 500kN servo MTS loading apparatus (Figure B.3.10) and loaded at a stroke rate of 1mm/3.5min, which was translated into individual strain rates for each sample length. During loading the strain gauge information and microseismic hits were recorded by the appropriate systems. In addition, the load and stroke
(axial deformation) were recorded with proprietary software owned by the Mining Department called TStress for Windows. This software used the sample geometry to calculate stress and strain using the load and stroke data, according to equations B.3.5 and B.3.6 (Figure B.3.9).

\[ \sigma_c = \frac{F}{A} \]  \hspace{1cm} \text{B.3.5}

\[ \varepsilon = \frac{d}{l} \]  \hspace{1cm} \text{B.3.6}

where \( \sigma_c \) is the UCS, \( F \) is the applied load, \( A \) is the surface area of the sample ends, \( \varepsilon \) is the strain, \( d \) is the stroke length and \( l \) is the sample length. At the end of the test the Young’s Modulus (E) was interpreted from the slope of the linear portion of the axial stress/strain curve (Figure B.3.9): \[ E = \frac{\Delta \sigma_c}{\Delta \varepsilon} \]  \hspace{1cm} \text{B.3.7}

Post failure, the sample was photographed and collected for thin section analysis (Figure B.3.11).

Figure B.3.9: Stress-strain curve from UCS testing of sample b023
Figure B.3.10: MTS 500kN servo loading machine used for UCS testing (Photo Courtesy of M. White)

Figure B.3.11: Photos of failed sample on MTS machine (left) and sample failure plane (right).
B.3.4.3 Data Processing

The first arrival times from the pulses were recorded for each microseismic phone and compared to its distance from the source to determine the p-wave velocity using equations B.3.8 and B.3.9.

\[ d = \sqrt{(N - N_0)^2 + (E - E_0)^2 + (Z - Z_0)^2} \]  

\[ v_p = \frac{d}{t - t_0} \]  

where \(d\) is the distance between the receiver and the pulsing phones, \(N\), \(E\) and \(Z\) are the northing, easting and \(z\) coordinates of both phones, the subscript 0 representing the pulsing phone, \(t\) is the first arrival time and \(t_0\) is the time delay between data recording and the pulse.

The average velocity was calculated based on the values from all six phones. In some samples the velocity varied considerably (+/- 20% in some cases) from the top of the sample to the bottom of the sample (for example Table B.3.1). This is likely due to the anisotropic p-wave velocity in the more foliated samples, and anisotropic sample damage due to the high in-situ stress state from which the samples were collected. The angle of p-wave travel in relation to foliation differs depending on the location of the sensor on the sample (Figure B.3.11), thereby resulting in different magnitudes of p-wave velocity at different locations.

![Figure B.3.12: UCS test sample schematic showing relationship between angle of p-wave travel and the foliation in anisotropic rock, which likely contributes to differing p-wave velocities at different receivers based on their location.](image-url)
Table B.3.1 Example p-wave velocities for sample b124, which has foliation defined by mica and feldspar grains oblique to the sample axis, and at varying angles to the direction of p-wave travel between the receiver phones and the pulsing phone.

<table>
<thead>
<tr>
<th>Phone</th>
<th>( v_p ) (( v_p ))</th>
</tr>
</thead>
<tbody>
<tr>
<td>1 (2)</td>
<td>5109 (4944)</td>
</tr>
<tr>
<td>3 (4)</td>
<td>3147 (3418)</td>
</tr>
<tr>
<td>5 (6)</td>
<td>2339 (2365)</td>
</tr>
</tbody>
</table>

The acoustic emission data were recorded by a microseismic hit tally software. Each time a microseismic phone sensed a wave above a threshold amplitude, the hitcount software added it to the tally. In most cases a rule-set requiring a minimum number of phones to record the same minimum amplitude event is used to record microseismic events, but the software is limited to four events per second, which at the loading rate used, would have resulted in a low data resolution and made interpreting the data difficult. Instead, the hits experienced by each phone were recorded and the maximum number of hits per second (on the order of \( 10^2 \) per second) experienced by 4 out of 6 of the phones was used as the event tally per second. These data were plotted versus time and versus stress and strain, and were used to identify initiation, interaction and propagation (Section 3.1), and compared to the numerical models in Chapter 4.

The load, axial deformation and strain gauge data were plotted as stress-strain (axial and lateral) and strain-strain curves. The axial deformation (giving average axial strain) was a more reliable measure of the axial strain than the values from the axial portion of the strain gauge, as expected. The lateral strain portion of the strain gauge gave equally poor data, nevertheless it is the only source of lateral strain data available for this sample size and rock type, and the graphs were used to compare to numerical model results in Chapter 4.

### B.3.5 Calibration of Point Load Test data with UCS

The UCS tests were conducted on samples belonging to the same dataset as the point load test samples described in Section B.3.3. Because of this spatial relationship, the UCS test values can be used to calibrate the point load test values. Two interpretation methodologies can be used to accomplish this: compare the UCS to the nearest few samples, say within 2-3m of the UCS test sample; or compare the UCS to all of the point load test results corresponding to the domain in which the sample falls. These domains are discussed in Section 7.1. All of the calibration was undertaken with the axial PLT results since that is the same loading direction as the UCS tests.
B.3.5.1 UCS Calibration to Nearest Neighbours

The calibration of UCS to PLT for nearest neighbours, for example samples within 2-3m of the UCS test sample, is a simple calibration method requiring the comparison of the UCS values to the PLT values. In highly variable rock types, however, it is difficult to select to which PLT value the UCS values should be calibrated. Table B.3.2 contains an example of UCS and PLT values over a 7m linear sampling distance. The selection of the correct calibration factor from such a variable set of results is not clear, although either an average could be taken (14.4) or the absolute nearest neighbours (+/- 1m) could be taken (17). Neither of these methods generates much confidence in the result, in particular since the rock domains for the samples at -3m, 2m and 3m distance from the UCS tests are different from the domain in which the UCS test is classified.

Table B.3.2: Example calibration factors from nearest neighbours to UCS test for sample b039.

<table>
<thead>
<tr>
<th>Distance from UCS Sample Location (m)</th>
<th>PLT (MPa)</th>
<th>UCS (MPa)</th>
<th>Calibration Factor (UCS/PLT)</th>
</tr>
</thead>
<tbody>
<tr>
<td>-3</td>
<td>8.97</td>
<td>12.1</td>
<td>12.1</td>
</tr>
<tr>
<td>-2</td>
<td>6.92</td>
<td>15.7</td>
<td>15.7</td>
</tr>
<tr>
<td>-1</td>
<td>5.40</td>
<td>20.0</td>
<td>20.0</td>
</tr>
<tr>
<td>0</td>
<td></td>
<td>108.26</td>
<td></td>
</tr>
<tr>
<td>1</td>
<td>7.72</td>
<td></td>
<td>14.0</td>
</tr>
<tr>
<td>2</td>
<td>10.02</td>
<td></td>
<td>10.8</td>
</tr>
<tr>
<td>3</td>
<td>7.77</td>
<td></td>
<td>13.9</td>
</tr>
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</table>

B.3.5.2 Calibration According to Domain Type

In highly variable rock, a more rigorous calibration method is by calibrating values according to domain type. As described in Section 7.1 the linear PLT data were classified according to geological features (fabric, mineralogy, macro variability) into domains. The domains into which the UCS test results fall are used, as a whole, to calibrate the PLT results within each domain to the UCS values.

A trial and error method was used to determine the best calibration factor for each domain as follows:

1. Each PLT result was multiplied by the calibration factor of 24.
2. A histogram and descriptive statistics of the trial calibrated PLT values were created for each domain.
3. The mean values were compared with the UCS values for each domain.
4. The calibration factor was increased or lowered, and steps 1-3 were repeated until the calibrated PLT mean was similar to the UCS values.

This method requires the assumption that the UCS value is representative of the mean UCS value for each domain. The results of this exercise give a calibration factor of 24 for all but three of the domains, for which the calibration factors of 22, 26 and 28 resulted in better fits. UCS data were not available for all domains, and for those domains with no corresponding UCS results, a calibration value of 24 was used, since it was the most applicable within the dataset and corresponds to the findings of Broch and Franklin {{241}}. The following are the UCS dataset, and descriptive statistics and histogram of the calibrated PLT values for the domains for which UCS values were available. Although the UCS tests suggest that the samples were stressed damaged, the comparisons stand since both the PLT and UCS samples were equally damaged.

B.3.5.2.1 Calibration of Domain B

Calibration factor = 22

<table>
<thead>
<tr>
<th>Sample</th>
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<tbody>
<tr>
<td>a153</td>
<td>70.24</td>
</tr>
<tr>
<td>a169</td>
<td>149.83</td>
</tr>
<tr>
<td>b011</td>
<td>135.7</td>
</tr>
<tr>
<td>b124</td>
<td>128.9</td>
</tr>
<tr>
<td>E-R-116700</td>
<td>170.9</td>
</tr>
<tr>
<td>E-L-116700</td>
<td>167.6</td>
</tr>
</tbody>
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<table>
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<tr>
<th>Parameter</th>
<th>Value (MPa)</th>
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<tbody>
<tr>
<td>Mean</td>
<td>126.0762</td>
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<tr>
<td>Standard Error</td>
<td>5.849626</td>
</tr>
<tr>
<td>Median</td>
<td>127.5291</td>
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<tr>
<td>Standard Deviation</td>
<td>48.94149</td>
</tr>
<tr>
<td>Sample Variance</td>
<td>2395.269</td>
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</table>

Figure B.3.13: Histogram of calculated UCS strengths using the calibration value
B.3.5.2.2 Calibration of Domain C

Calibration factor = 24

<table>
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<tr>
<th>Sample</th>
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</thead>
<tbody>
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<td>a065</td>
<td>150.442</td>
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<td>Median</td>
<td>131.4372</td>
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<tr>
<td>Standard Deviation</td>
<td>50.03119</td>
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<td>Sample Variance</td>
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Figure B.3.14: Histogram of calculated UCS strengths using the calibration value

B.3.5.2.3 Calibration of Domain E

Calibration factor = 28

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<td>b039</td>
<td>108.26</td>
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<td>E-R-116600</td>
<td>131.5</td>
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<tr>
<td>E-L-116600</td>
<td>168.3</td>
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<td>Standard Deviation</td>
<td>61.92741</td>
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B.3.5.2.4 Calibration of Domain G

Calibration factor = 24

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<td>E-L-116650</td>
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</table>

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</thead>
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<td>Median</td>
<td>190.8284</td>
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<td>Standard Deviation</td>
<td>47.96444</td>
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<tr>
<td>Sample Variance</td>
<td>2300.587</td>
</tr>
</tbody>
</table>

Figure B.3.15: Histogram of calculated UCS strengths using the calibration value

Figure B.3.16: Histogram of calculated UCS strengths using the calibration value
B.3.5.2.5 Calibration of Domain I

Calibration factor = 24

<table>
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<td>c106i</td>
<td>68.3</td>
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<td>c106ii</td>
<td>105.02</td>
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<table>
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<th>Value (MPa)</th>
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<td>Standard Error</td>
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<td>Median</td>
<td>70.12378</td>
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<td>Standard Deviation</td>
<td>25.11059</td>
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<td>Sample Variance</td>
<td>630.5416</td>
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</table>

Figure B.3.17: Histogram of calculated UCS strengths using the calibration value

B.3.5.2.6 Calibration of Domain J

Calibration factor = 26

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<th>Sample</th>
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<td>E-R-116800</td>
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<td>E-L-116800</td>
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<table>
<thead>
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<th>Value (MPa)</th>
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<tr>
<td>Mean</td>
<td>107.1619</td>
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<td>Standard Error</td>
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<tr>
<td>Median</td>
<td>108.0675</td>
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<tr>
<td>Standard Deviation</td>
<td>38.54491</td>
</tr>
<tr>
<td>Sample Variance</td>
<td>1485.71</td>
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</table>
Figure B.3.18: Histogram of calculated UCS strengths using the calibration value
Appendix B.4 : Geological Record of Southern Aar Granite Between Tunnel Metre 116428 and 116812

B.4.1 Introduction

The geological data were amalgamated into a single linear figure according to tunnel metre location. The side maps, rock mass domains were digitised from records taken from inside the tunnel. The point load strength testing data were digitised as a smoothed line joining all points. The F Factors were collected from the point load strength testing samples on 2m intervals, when sufficient sample matter was available. Each of the F Factors described in Chapter 4 were attributed a category: high, medium or low according the impact each value was considered have to spalling sensitivity. The spalling sensitivity, and spalling sensitivity with fabric were calculated from the F Factors and the values were digitised. TBM performance data were digitised using the NAR and DI values, as well as categorised according to the relationship between NAR and DI, locations with face instability were noted with a start. The depth overbreak at the tunnel wall and tunnel face were also digitised. This Figure is uncalibrated and consists of raw F Factor data compared to indicators of TBM performance and tunnel stability, as well as rock mass properties and laboratory strength testing values. This Figure is found on the DC accompanying the thesis.
Appendix C

This appendix contains documents related directly to the material presented in Chapter 3.
Appendix C.1 : TBM Performance Data

C.1.1 Introduction

TBM performance data were obtained by a data acquisition system (DAS) operated by Herrenknecht AG. The details of the DAS are not available and will not be discussed here. The TBM data are formatted into a spreadsheet containing entries for equipment status every 10 seconds. Each individual spreadsheet file represents one stroke of the thrust pistons, roughly 2 m advance distance. The majority of the data contained in the files is not used, but the data representing thrust, torque, RPM, speed, penetration rate and the surveyed location of the head based on the extension of the thrust pistons comprises the data used throughout this research.

The data in the files is raw, meaning that it is directly input from the instruments that monitor the status of the equipment, except for speed and penetration rate, which are calculated by the data acquisition system. Speed is a velocity calculation based on the piston extension length change over the 10 second interval of data acquisition. Penetration rate is the speed normalised by the RPM, giving the distance covered during one revolution of the TBM head. Due to the vast amount of data represented by these TBM files, average stroke values were calculated to ease data analysis and performance comparison to geological conditions. The following sections present the approach to average creation, justification for the methodology and assumptions made.

C.1.2 Average Stroke Data

The data acquisition system (DAS) is capable of generating average stroke data, but the methodology used by the DAS contains some inadequacies for research applications addressed in the macros developed for this research. In the DAS average calculation the values recorded during active advance (when the TBM is pushing forward) are used, resulting in the inclusion of values from start-up, including zero values for penetration rate. In the macros developed for this research, data are cut-off when thrust drops below a certain value (4000 kN for the Amsteg machines, for example). This removes the impact of start-up and slow-down from average driving values. By removing these values, the average values more appropriately represent true TBM driving conditions, which can then be used to make interpretations relating to geological conditions.
The penetration rate, thrust, torque, speed and RPM are calculated by this methodology: the macro sums all data entries corresponding to a thrust greater than the threshold, once it reaches the end of the file it divides the sum by the total number of entries summed. A second method for calculating the penetration rate was based on Weh (2003 personal communication) and relates the distance covered over the stroke to the total number of revolutions undertaken during the stroke. This requires accurate RPM values (not always possible in the Raron/Steg dataset) and accurate piston extension measurements, which is not always possible. Regardless, both penetration rate values are typically very similar and are used to identify glitches in the dataset when they do not match.

The average data are saved in a central spreadsheet for each rock type, and from there are used for TBM performance analysis.

**C.1.3 Net Advance Rate**

To address issues of TBM performance impact due to face instability, a new TBM performance parameter was created using the raw stroke data. The net advance rate (NAR) was designed to highlight face instability induced advance rate reduction during a stroke by employing knowledge about the manner in which the TBM is operated in unstable face conditions. At the Amsteg worksite the operation policy was to monitor torque requirements during excavation through potential face instability conditions. If the torque increased suddenly above a 30-40% of maximum torque threshold, the thrust was stopped, but the machine head was continued to rotate. The aim was to minimise muck entry into the buckets during face failure and reduce the potential for damage to the machine head of the muck conveyor system. Once the muck was removed, active excavation (using thrust) could resume. In conditions of frequent face failure, this process was repeated several times during excavation, resulting in slower overall advance rates arising from geological conditions unfavourable for face stability.

The NAR was calculated for all strokes, regardless of face stability conditions in an attempt to highlight domains of face instability leading to increased or decreased TBM performance. In addition to highlighting areas and magnitude of face instability impact, the NAR also quantifies the impact of stable rock that is tough to excavate. The NAR is, therefore, a measure of the advance rate (or speed) of the machine related to conditions at the face. In the research, the NAR is compared to average speed to highlight locations where they deviate (suggesting face instability) and locations where they are comparable (suggesting stable face conditions). In most analyses, NAR is used instead of speed as it is a more encompassing
measure of TBM performance associated to geological conditions and their impact on TBM excavation.

The NAR is calculated as the total distance covered in the stroke divided by the active TBM driving time. The active TBM driving time is defined as the time during which the TBM head is turning (i.e. when RPM is non-zero). This removes non-face related impacts to the advance rate during a stroke, such as additional support time requirements, automatic machine stops due to metal on the conveyor system or malfunctioning equipment, machine stops for maintenance, etc. The macro calculates the stroke length and sums the driving time during which the condition for non-zero RPM is true, the quotient of which is the NAR.

The NAR is calculated for all datasets (Steg/Raron, Amsteg and Bodio) with the assumption that unstable face conditions are driven using the same procedure as the one used in Amsteg. Once the NAR is calculated, it is input into the central spreadsheet containing all other average data.
Appendix C.2 : Combination of Dataset Locations

C.2.1 Introduction

Three very important datasets were collected during field work: TBM performance data, overbreak records and point load test (PLT) data. All three datasets are correlated to the tunnel by the tunnel metre location, but the locations contained within the datasets are not comparable to each other. In order to integrate the datasets a macro was written to associate the PLT data and overbreak records to the TBM data locations.

C.2.2 Point Load Test Locations

The point load tests were conducted on drill core collected ahead of the tunnel face. The individual samples were numbered according to their distance from the beginning of the drill core, which was located within the tunnel by a tunnel metre location. Using this information and knowledge of the plunge of the drill core the tunnel metre location of the PLT samples was estimated. Unfortunately, the drill core was not perfectly straight and at one point reversed its plunge and was intersected in the face, making the tunnel metre locations very rough estimates. Additional samples were collected from the tunnel face or the conveyor belt during excavation and accurately located. By comparing these samples to the PLT samples in the approximate tunnel metre locations a more accurate estimate for their tunnel metre location was made. This was further aided by comparing rock mass types in the PLT dataset to the side maps collected during field work. In most cases the locations deviated from 1 to 3 metres, which was adjusted in the datasets.

C.2.3 Relation of Point Load Test, Overbreak and TBM Datasets

The point load test (PLT) samples were collected at 1 metre intervals, while the TBM performance data were collected at approximately 2 metre intervals. To relate the two datasets a macro was written that related averaged PLT to midpoint of each TBM stroke. Since the frequency of data in the point load dataset is higher than in the TBM performance dataset, steps were taken to average point load data points falling within a stroke. For each TBM stroke the tunnel metre location was recorded, the PLT data within 1.1 metres of this location were averaged
and assigned to that tunnel metre location. The reason 1.1 metres was selected is that the strokes are separated by approximately 2 metres, and this spacing ensures that a PLT value is assigned to each TBM stroke location.

The overbreak records include tunnel wall and face overbreak, both of which were assigned to TBM stroke locations. A similar method was used as for the PLT data points, where overbreak locations were assigned to the nearest TBM stroke location. The data were typically more than 2 metres apart and no averaging of values was necessary.
Appendix D

This appendix contains documents related directly to the material presented in Chapter 4.
Appendix D.1: GEAT05 Symposium Manuscript

This appendix contains a paper presented at the GEAT05 Symposium in Zurich, September 2005, and published as a chapter in the symposium proceedings.
Effects of Mineralogy and Tectonic Deformation History on TBM Advance in Deep Tunnelling

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2 MIRARCO, School of Engineering, Laurentian University, Sudbury, Ontario, Canada
3 Herrenknecht AG, Schwanau, Germany

ABSTRACT: The prediction of TBM penetration in hard rocks is typically based on conventional rock strength properties, such as UCS, standardized penetration and abrasion tests, and to some extent, rock mass characterisation. In order to improve TBM performance prediction, an approach to data collection, management and analysis has been developed to investigate the effects of mineralogy and macro- and micro-tectonic properties of rock on the excavatability. A system for on-site geological and machine data collection was developed to obtain a dataset with which the rock-cutter interaction process can be investigated. This dataset, consisting of rock samples, geological maps, digital photos, thin section images, classification tables, TBM machine data and rock testing properties, was organized by tunnel location to permit an integrated process of investigation. A petrographic and microtectonic classification system was developed to relate composition, texture and the presence and intensity of key fabric elements to the observed tunnel machine performance. Combined with fundamental knowledge of rock mechanics and material properties, this classification has been refined to obtain practical indicators for the influence of microproperties on TBM excavation. The adopted approach and some preliminary results are presented.

D.1.1 INTRODUCTION

A number of TBM performance prediction techniques for hard rocks have been developed based on empirical data (i.e. Bruland 1998), scale lab testing (i.e. Rostami et al. 1996), rock mass classification (i.e. Barton 2000) or index tests (i.e. Dollinger et al. 1998). Few studies, however, have been conducted to relate TBM performance to geological properties of hard rocks through fundamental mechanisms of the rock fragmentation process (Büchi 1984). Typical input values into current predictions systems include conventional strength tests values, such as UCS, rock mass classification values, and a selection of index test values, such as CERCHAR (CERCHAR 1986) abrasiveness index, designed to evaluate the hardness and abrasiveness of a rock.
Gertsch (2000) investigated rock toughness as it relates to disc cutting by means of linear cutting tests and strength tests, and the results were described in terms of observable outcomes, such as specific penetration and debris yield. The conclusions suggest that in order to improve TBM performance prediction, further investigations should relate the physical properties of rocks and the mechanics of rock failure.

Methods for analyzing the effect of geological characteristics of rocks, such as anisotropy and mica content (Büchi 1984, Shea et al. 1993, for example) and microfractures (Nasseri et al. 2005, Frederich et al. 1990, Brosch et al. 2000, for example) on rock strength have been developed. Little consideration is currently given to the tectonic history of the rock (for example Göransson et al. 2004), which can be used to understand the origin of geological characteristics. Including this information for TBM performance prediction might be very useful in cases where direct laboratory testing of specimens is sparse or cannot be conducted, especially when new rock types are encountered. Such is the case for the deep Alpine tunnels that make up the Neue Eisenbahn-Alpentransversale (Neat) project in Switzerland and forms part of this investigation. The current research is carried out in the Gotthard tunnel, Switzerland; in the lots excavated by Herrenknecht hard rock open TBMs, each with a diameter of 8.83 to 9.58 m.

D.1.2 GEOLOGICAL SETTING

The Gotthard base tunnel passes through five units that have undergone overprinting or metamorphism during the Alpine Orogeny (Trümpy 1999). These include the northern, central and southern parts of the Aar Massif, the Tavetsch intermediate Massif, the Gotthard Massif and the Lucomagno and Leventina gneisses (Fig. D.1.1). Four rock units made of sedimentary and volcanic-sedimentary cover sequences are found between the five crystalline units and include the Intschi zone, the Disentis zone, the Urseren-Garvera zone and the Piora synform (Trümpy 1999).

The rocks of the Aar and Gotthard Massifs, and the Lucomagno and Leventina units, particularly the domains that contain massive to moderately fractured or sheared (GSI >70) rock, are of interest because their intact rock properties, rather than distinct large scale features, govern TBM performance. The rock units discussed in this paper are the Altkristallin rocks of the Aar Massif in the north and the Leventina gneiss in the south.

The Altkristallin unit is significant for investigating macro scale deformation (cm scale foliation) effects on excavatability. This unit is part of the Aar Massif, which is a collection of intrusive and metasedimentary rocks of varying ages that was thrust 25-50km during the advancement of the Alpine orogeny (Burkhard 1999). The Altkristallin are complex pre-
Variscan basement rocks that have undergone contact (intrusion of Aar granite) and regional (Variscan, Alpine) metamorphic episodes and consist of highly foliated to migmatitic, granitic to tonalitic gneisses (Abrecht 1994).

The Leventina unit, in locations where it is not affected by brittle deformation features, is hard, massive granodioritic gneiss that has been metamorphosed to amphibolite facies (Zappone et al. 1996). The basement of the Penninic domain, to which the Leventina unit belongs, was subducted during the advance of the Alpine orogeny and subsequently uplifted, perhaps by denudation, as the orogeny progressed (Beaumont et al. 1996). This may account for the amphibolite facies deformation experienced by the Leventina unit, suggesting high temperature and pressure deformation. The portions of Leventina gneiss that do not exhibit macro-scale brittle deformation features (i.e. large shears and faults) are much stronger, and are therefore of interest to this research.

D.1.3 MATERIALS SCIENCE AND TECTONICS

The goal of the current research is to relate TBM performance to geological properties of hard rocks, using materials science and tool-rock interaction as its basis. In the most efficient excavation method, fractures are induced by crushing within or adjacent to mineral grains at the cutter tip under medium confinement (Fig D.1.2). Dilation generated during this crushing process, as well as additional fracture nucleation and extension, drive macroscopic crack propagation under low confinement parallel to the excavation and away from the cutter contact to form chips (Fig D.1.2). Different characteristics of the mineral and grain assemblage as well as internal deformation features (high confinement) have an impact on the material behaviour.

The geological controls on the fracturing mechanism are a consequence of the deformation incurred during the tectonic history (i.e. shortening, extension, subduction, etc. with or without heating), and include: at the macro scale indicators of high strain, such as foliation, shears, and faults; at the micro scale, grain alignment, grain scale and shape change, and grain scale deformation (Nicolas and Poirier 1976). Figure D.1.3 shows examples of micro scale plastic deformation features from the Leventina gneiss.

Figure D.1.1. Cross section through the major units of the Gotthard tunnel (after Keller 1999). Vertical scale same as horizontal scale.
Myrmekitic texture, mechanical twin formation and bending all suggest that the rock has undergone significant strain. Grain size variability, such as the presence of porphyroclasts and fine recrystallised or crushed matrix are evidence of the strain and thermal history. Evidence for subgrain formation, grain boundary migration and recrystallisation are indications of increasing strain and rate of heating and cooling.

These features are comparable to features induced during plastic deformation and recovery (annealing) of metals. In the case of metals, one can cold work a material or anneal it to increase or reduce its strength (Illston et al. 1979) while in rocks these processes occur naturally by tectonic deformation during orogenic events. By cold working a metal, one will induce formation and movement of dislocations, defects in the crystal lattice. It is the
accumulation and, more importantly, the interaction of dislocations that strengthens a metallic grain, by making it more difficult for plastic strain to occur through dislocation glide since dislocations tend to repel each other and block each other’s movement. Conversely, recovery will reduce the strength by allowing the dislocations to rearrange and annihilate themselves (Illston et al. 1979).

Figure D.1.4a shows how “cold working”, or deformation at temperatures well below the recrystallisation temperature, can increase the peak strength of a grain but reduce the ability to accommodate plastic strain, making it more brittle. Figure D.1.4b shows how annealing (high temperature treatment) will induce recovery (movement of dislocations), recrystallisation and finally grain growth with increasing temperature, resulting in decreased strength and increased ductility. In rocks these processes occur separately only in rare occasions, so the features of both deformation and recovery will be present, in different concentrations depending on the tectonic history and mineralogy.

A consequence of this in polycrystalline materials may be that they behave dissimilarly to metallic materials. Hacker and Christie (1990) suggest that a lack of dislocations may lead to increased brittleness, the opposite of what is thought of for metals. Their work with amphibolites (containing plagioclase and some quartz) showed that samples with signs of recovery and dynamic recrystallisation behaved more plastically than samples without such signs (Hacker and Christie 1990). Unravelling the effect of dislocation nucleation and movement during plastic deformation on polycrystalline material strength, compared to well documented processes in metals, will be critical for relating geological history to TBM performance. Much work relating tectonic deformation to rock strength has been conducted at higher temperature and lower strain rates (Passchier and Trouw 1996, Hacker and Christie 1990, Göransson et al. 2004) than encountered at the TBM head and applying these concepts to mechanical excavation environments will also be a critical step in this investigation.

Figure D.1.4. Effect of deformation and recovery on metals: a) Cold working of Carbon Steel (after ASM 1990); b) Annealing of a brass alloy (after Sachs and Van Horn 1940)
D.1.4 DATA COLLECTION

D.1.4.1 Field Investigation Program

The selection of data collection sites was dictated partly by logistics, and partly by geology. Since the focus of this research is TBM excavation of hard rocks, data collection was undertaken only in mechanically excavated tunnels through hard rock and not undertaken in locations where the TBM encountered rock with large-scale brittle deformation features, such as those encountered intermittently in the Leventina gneiss. Large scale brittle deformation features, including joints and shear zones, will dominate the mechanical behaviour of a rock, and would overshadow any microtectonic controls, which are of most interest to this thesis. The rocks encountered at the North and South ends of the Gotthard tunnel were thus selected for this project. Despite its applicability to this research, no data collection was undertaken in the Gotthard massif, since the TBMs have not reached this tectonic unit at the time of this study.

D.1.4.2 Geological Data Collection Methodology

A system integrating the onsite logistics and the time necessary to collect data underground was designed and modified according to the worksite. The linear nature of this project mandated a system with the ability to cross-reference all data sources with known tunnel locations. All data collected were organised by tunnel metre location in a database developed specifically for this project to simplify data manipulation.

The shape of the tunnel was a determining factor in the mapping methodology design. For a linear tunnel, the mapping was best organised in sequential, linear mapping sheets (Fig. D.1.5d). Only the top third (which had easiest access from the TBM) of the tunnel was mapped to ease the mapping, and to minimize interference with the machine operation. In addition both geological and geotechnical information was recorded on the mapping sheets, in separate linear areas for recording geological (rock type, fractures, foliation, etc.) and behavioural (spalling, bursting, wedges, etc.) information. Detailed geological information was also collected to complement the tunnel scale mapping by means of specially created detailed description sheets (Fig. D.1.5a). To allow for a three-dimensional representation of failure features a cross-section map sheet was created (Fig. D.1.5b).

Digital photos of the tunnel wall were taken regularly (Fig. D.1.6) to complement the mapping sheets. Close-up (macro) and tunnel scale digital photos were also taken with each detailed geological description location to show the mm to cm scale features of the geology, such as phenocrysts, grain alignment or small scale fracturing, and the context in which the features taken in the macro photos lie in the rock, respectively.
The rock sampling methodology occurred at frequent intervals subject to subsequent shipping limitations. Sampling frequency depended on frequency of geological change and availability of sampling locations, and occurred roughly every 4 to 8 m. Large enough samples were selected for thin sectioning and, where possible, for later rock testing. Samples were annotated with location and orientation information.

Consistency in data collection methodology was essential to ensure comparability. Additional notes and photos were correlated either to tunnel metre location at which the tunnel wall behaviour was noticed (i.e. location of a large spall or dramatic change in geology) or the tunnel metre location of the TBM head at which the excavation behaviour was noticed (i.e. muck type on the conveyor, vibration levels of the machine).

TBM performance data for the various rock types were necessary to compliment the geological data collected using the methodology described above to fulfill the goal of the research. A set of TBM stroke data spreadsheets corresponding to the locations where mapping was conducted was collected for analysis.

![Figure D.1.5. Examples of: a) Detailed geology description; b) cross section sheet; c) thin section description; d) mapping sheet](image-url)
D.1.5 THIN SECTION ANALYSIS

D.1.5.1 Sample Selection and Thin Section Preparation

A major component of the laboratory analysis of the data collected onsite included examination and classification of rock samples by transmitted light microscopy. For this research, sample selection is based on the following factors: proximity to other thin section locations, rock type, macro scale features, tunnel wall behaviour and TBM performance. The larger samples were kept for later strength testing. General orientation of the thin section with respect to cardinal axes or the rock fabric was sufficient for this research. Samples from areas where macro scale features, such as intense fracturing or pervasive foliation, were dominant were not selected for thin sectioning.

D.1.5.2 Thin Section Classification

A system for recording geological features from thin sections was developed based on the detailed geological description sheets (Fig. D.1.5c) to obtain information regarding tectonic history. In addition to mineral content and grain size, microtectonic features that are visible under transmitted light microscopy were recorded. These microtectonic features include those that indicate more plastic (undulose extinction and subgrain boundary formation, etc.) and less plastic (i.e. cataclasis and microfracture of grains) deformation processes and features that indicate plastic recovery processes (grain boundary migration, recrystallisation, etc.). Excavation induced fractures were also examined as they can be used to understand the fracturing process of these rocks during excavation.

The goal of the thin section analysis is to compare rocks with similar dominant mineralogy (quartz, feldspar, micas) but different physical characteristics (fabric, grain size and orientation, microtectonic deformation) to investigate the effect of different tectonic histories on TBM performance. The research is ongoing and only a few samples have been investigated in detail. These samples were used in the development the thin section classification methodology, where indicators of TBM performance were used to select rocks with different performance to investigate their different geological characteristics.
Figure D.1.6. Macro scale photos (top) and tunnel scale photos (bottom)

Figure D.1.7. TBM performance indicator (ratio of penetration to gross TBM thrust) with moving average and sample locations (triangles) used to compare performance of TBM through changing rock types within the Altkristallin. A: low performance in rock with high crushed versus relict grain ratio; B: average performance in mildly plastically deformed granitic rock; C: high performance in rock with larger percentage and higher degree of contact of relict grains.

Figure D.1.7 shows an example of the approach used in the Altkristallin. The rock mass is cohesive and has undergone a complex polymetamorphic history related to at least two orogenic events (Abrecht 1994), resulting in mineralogical and physical characteristics related to both brittle (i.e. cataclastic grain size reduction) and ductile (i.e. flow of the fine grained matrix resulting in defined foliation) deformation processes.
D.1.6 PRELIMINARY RESULTS

The preliminary results suggest that for hard, intact and non-foliated rock the competition with respect to rock strength seems to be between several features of tectonic deformation and recovery. While micro scale plastic deformation processes generally strengthen a metallic material by inducing dislocations in the crystal lattice (Illston et al. 1979), the smaller grain size resulting from recovery, in particular dynamic recrystallisation (Nicolas and Poirier 1976, Passchier and Trouw 1996), of deformed grains will tend to strengthen it. Several factors are clearly involved and are not yet fully understood, however early analysis of thin sections and comparison to the TBM performance is providing important data on some of the interactions.

In Figure D.1.8 three rock types are compared to their TBM performance at low thrust. The non-foliated rock types at centre and right are more difficult to bore than the foliated rock at left. The textures in the non-foliated rocks are vastly different where the rock labelled as ‘medium’ is equigranular with straight grain boundaries, while the rock labelled as ‘strong’ has larger grains with very interlocked grain boundaries and definite signs of strain, such as mechanical twins and grain boundary migration.

Analysis of fractures induced by the TBM or wall stress in massive unfoliated ground shows that fractures will propagate around some grain types and through other grain types. Identifying the geological controls, such as mineralogy, grain size and shape, and accrued plastic (ductile deformation and recovery) and brittle (cataclasis and microfracture) micro-tectonic deformation, on fracture propagation is a key focus of this research.

Figure D.1.8. Schematic comparing gross thrust and resultant penetration rate during TBM start-up. The shape and slope of the line appears to be related to the geology and can be used in comparing geological characteristics to machine performance. For example, crystalline rock that is progressively more difficult to penetrate corresponding to A: foliated, B: equigranular and C: larger grained, highly interlocked.
For cohesive foliated rock, fabric features incurred during brittle and ductile shearing (Chester et al. 1985, Passchier and Trouw 1996), such as orientation and alignment of grains, percentage of relict grains versus crushed, sheared or recrystallised matrix, and degree of grain size reduction due to brittle (such as cataclasis) and ductile (mylonitisation) processes likely play a major role on strength of the rock (Passchier and Trouw 1996) and efficiency of fracture propagation during excavation. The behaviour of such rock under compressive stress (at the cutter tip) and extension stress (along the tunnel wall) can be quite different and can result in an easily excavated rock (Fig. D.1.8) that exhibits little wall damage. A non cohesive foliated rock, such as a rock with high frequency of macro fractures or distinct non-cohesive shear zones, would exhibit high excavation rates but poor tunnel wall behaviour due to ravelling and wedge failures, and is not a focus of this research.

These results, while preliminary, give direction for modification of rock strength classification to account for tectonic effects. The goal is not to introduce thin section analysis for TBM design but rather to design a simple classification system relating tectonic history (for example, deformation under low or high temperature conditions and subsequent recovery) to rock strength, based on the analyses of different rock types.

D.1.7 CONCLUSIONS AND FUTURE WORK

Constrained data collection and laboratory analysis methodologies were developed to obtain and analyse data from a linear tunnelling project. The Gotthard tunnel crosses several structural domains and rock units within these domains. This research focuses on massive, hard rock tunnelling, limiting the rock types that are of interest for data collection and analysis to the crystalline basement rocks of the Aar and Gotthard Massifs and the Penninic Basement gneisses of the Lucomagno and Leventina units. In addition, the underground nature of the field site limits the data collection to those sites that are exposed during data collection, namely the Aar Massif and the Leventina gneiss.

The data collection methodology was designed to maximise the data collection in the available time. To ensure that comparable data was collected and that as little time as possible was spent processing the data, data collection sheets were created beforehand for linear tunnel wall mapping, detailed geological information collection and cross section recording and were input into a cross-referenced sample and photo recording system. A thin section analysis methodology was developed for quantitative geological interpretation. Investigation sheets were used to record the microtectonic features that would indicate strengthening or weakening of the intact rock, for comparison to TBM performance data collected onsite.
The preliminary results from the thin section analysis show that classification of rock as strengthened or weakened due to tectonic deformation and history can be complex. The key is to better understand the competing processes that occurred during geological history, including shortening, subduction, uplift, shearing, faulting, etc. that could affect the intact rock strength.

Other work related to the data collection and analysis includes numerical modelling of the fracture process under disc cutters. The data collected onsite will be used to provide information regarding the strength of intact rock and the effect of anisotropic properties, such as foliation or grain alignment. Future work will also include investigation of fracture processes at the micro scale and the effect of grain mineralogy, shape, deformation and recovery processes and the presence and nature of original microfractures on initiation and propagation of microfractures induced by the excavation process.

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REFERENCES


Appendix D.2 : Thin Section Data Collection
Methodology

D.2.1 Introduction

A total of 230 thin sections were selected, prepared and analysed for this research. The thin sections were from samples collected at the Raron/Steg worksite of the Lötschberg tunnel, and the Bodio and Amsteg worksites of the Gotthard tunnel. Thin section analysis was selected because it is the only reliable but fast and simple method for petrographic and textural analysis. Thin sections consist of a very thin slice of rock mounted on a glass microscope slide and covered with a slide cover. A transmitted light microscope is used to observe the thin sections through which light will pass and allow the identification of individual grains, mineral types, texture and fabric.

D.2.2 Sample Selection and Preparation

The samples collected from the various worksites in the Lötschberg and Gotthard tunnels (Appendix B.3) were compared to the available TBM data and any strength testing results. Locations in which the TBM performance data suggested that the rock was behaving in different ways, or in ways that were interesting to this research were selected and the nearest (within 1-3m) sample was selected for thin section analysis. Once these samples were analysed and their results were interpreted, it was decided to increase sampling frequency within the dataset for which one sample is available at each tunnel metre, and for which point load testing was performed.

Once the samples were selected, they were trimmed to expose the surface from which the thin section would be cut, these are called chips. The thin section surfaces were aligned perpendicular to foliation and parallel to lineation, in rocks with fabric, such that the fabric could be best characterised. The chips were also trimmed to the size of a microscope slide, and labelled for thin section preparation. Thin sections were prepared by an expert thin section technician employed by the Department of Geological Sciences and Geological Engineering.
**D.2.3 Thin Section Analysis**

Thin sections were analysed using a transmitted light microscope with quartz plate and crossed polarisation lenses. The crossed polarisation lenses allow for grains and most mineral types to be identified, Figure D.2.1, although the plane light is useful for identifying biotite, chlorite, amphibole and opaque minerals in particular, due to their colour under plane polarised light. Feldspar and quartz can only be differentiated from each other, for the most part, under crossed polarised light, since both are usually clear under plane light. Microstructures, such as subgrain boundaries and twinning are also only visible under crossed polarised light.

During thin section analysis of the first dataset of thin sections, each chip cut surface was scanned, then each sample was photographed through the microscope, its mineralogy, grain size and fabric were recorded, special characteristics, such as fractures, fabric, or uncommon minerals, etc., were noted and several representative sketches were made (Figure D.2.2). For thin section analysis of the second, higher sampling frequency dataset only a petrographical and fabric analysis were conducted to expedite the process.

![Photomicrographs of samples GA_a045 (Amsteg) and GB_3956 (Bodio), top and bottom respectively, showing the same location under plane light (left) and crossed polarised light (right).](image)

Figure D.2.1: Photomicrographs of samples GA_a045 (Amsteg) and GB_3956 (Bodio), top and bottom respectively, showing the same location under plane light (left) and crossed polarised light (right).
D.2.4 F-Factors Database

A MS-Access database was created to collect the thin section analysis data. This analysis comprised of the collection of petrographic and fabric data to be used for geomechanical characterisation. The database form in Figure 8.7.3 was used to input the raw data into the database. Scripts written into the form calculate the F-Factors related to the combinations of

Figure D.2.2: Sketches of thin sections clockwise from top left: GA_095, GA_102, GA_111, LS_31.
geomechanical parameters entered into the form. These data were output to excel for further manipulation and interpretation.

Figure D.2.3: Screen capture of the data collection form for F-Factors.
Appendix E

This appendix contains documents related directly to the material presented in Chapter 5.
Appendix E.1 : Numerical Methods

E.1.1 Numerical Modelling Code Types

There are three basic types of numerical modelling codes used in geotechnical applications: continuum, discontinuum and hybrids, the selection of which depends on the behaviour of interest (Jing and Hudson, 2002) (Figure E.1.1). Continuum codes are used for intact rock or a homogeneous rock mass, which may contain discontinuities that are taken into account in the parameter selection for the rock mass. Discontinuum codes are used for rock masses where behaviour along discrete discontinuities dominates over behaviour of the intact rock, and where the discontinuities cannot be simplified into a homogeneous rock mass. Hybrid models are used for rock masses where behaviour in the intact rock and at the discontinuities is important at some time during the simulation.

For continuum applications, three basic numerical methods exist: finite difference, finite element and boundary element. Boundary element methods, such as Examine 2D and Examine 3D (Rocscience Inc., 2005), are comprised of a set of interior boundary conditions (for example outlining an excavation) and an infinite assumption for the exterior boundaries, making them very useful for investigation of stress conditions and displacements using elastic properties (Jing and Hudson, 2002). Finite element methods, such as Phase2 (Rocscience Inc., 2005) and ABAQUS (ABAQUS Inc.), use a set of differential equations to represent the physics in the model, which are then converted to a matrix of linear algebraic expression, the constituents of which are adjusted to minimise error or energy terms (Carter et al, 2000), with displacements and yielded elements as the output, from which stresses and strains can be calculated (usually by the model). The system is solved implicitly in one step, or can be broken down into smaller steps, using trial functions to approximate the solution (Jing and Hudson, 2002). The grid can be irregular, but large displacements and nonlinearity may lead to instability in the system and cause it to crash, rendering the results prior to the crash meaningless.
Finite difference methods, such as FLAC and FLAC3D (Itasca Consulting Group Inc., 2007), use an explicit iterative process to incrementally solve the discretised governing partial differential equations at finite time steps. The equations and their solutions are directly and locally solved, making this process straightforward for complex constitutive material properties, and requiring less computer memory (Jing and Hudson, 2002). Non-linear and large displacement problems are easily solved, but may require numerous timesteps to prevent instability (Carter et al, 2000), and irregular mesh geometries have been made possible by Voronoi tessellation and polygonisation capabilities in current configurations (Jing and Hudson, 2002).

**E.1.2 Fast Lagrangian Analysis of Continua (FLAC)**

For the purposes of this research, continuum methods are sufficient to investigate the behaviour of massive and intact rock under high stress but low-confinement loading (schematic at far right in Figure E.1.1), while the finite difference formulation is well suited to modelling the fracturing process of heterogeneous strain-softening constitutive models with moderate to large displacements. The numerical modelling code FLAC (Itasca Consulting Group Inc., 2007) was selected because of its formulation for rock mechanics modelling and its applicability to the research.

FLAC uses a Lagrangian grid formulation in which the gridpoint displacements are calculated during each timestep and the coordinates are updated only at the end of the timestep, thereby making the calculation fast; this means that the locations and velocities of gridpoints are taken as constant within the timestep. For the model to remain valid, the timesteps must be taken to be small enough that constant locations and velocities are a physically justified assumption based on the elastic properties of the material. The strains at each timestep are, therefore, small,
but the cumulative strain over several timesteps can be large (Itasca Consulting Group Inc., 2007).

The timestep marching scheme in Figure E.1.2 illustrates the sequence employed by FLAC. At each timestep, $\Delta t$, the equations of motion are invoked to obtain new accelerations, velocities and displacements from applied forces at the nodes. The shape function relates the gridpoint displacements to the element shape, outputting the resulting strains within the element, which are then converted to stresses within the element through the constitutive model equations. Finally the new forces at the gridpoints are calculated by relating the stress within the element to its area and shape. At this point the gridpoint locations are updated and the cycle will begin again with the new applied forces at the gridpoints. Equilibrium is not necessarily sought by FLAC and it is up to the user to determine the number of timesteps required to satisfactorily achieve equilibrium (Carter et al, 2000). The size of the timesteps is determined by FLAC based on the period of the system being modelled, but the user can also specify a timestep size.

The equations of motion are set in FLAC, but the constitutive model input by the user will affect execution of the left portion of the cycle in Figure E.1.2. As the choice of constitutive model and the input parameters are critical to this research, a discussion regarding the selection and calibration is found in Section 4.2. The specific mathematical formulations used within the FLAC timestep cycle can be found in the FLAC manuals (Itasca Consulting Group Inc., 2007), and more information regarding numerical modelling methods can be found in Carter et al. (2000) and Jing (2002).
Figure E.1.2: Schematic computational cycle used in FLAC showing the physical equations used during the cycling (modified from (Jing and Hudson, 2002).

### E.1.3 Library of FISH Functions in FLAC

A programming language was developed by Itasca to allow users to create new variables and functions in any of the Itasca codes. This language is called FISH and is a compiler embedded into the FLAC code, which enables the implementation of new variables, user-defined grids or loading schemes, the automation of FLAC procedures, output variables or element states, and basic programming functions such as loops, if statements, and queries (Itasca Consulting Group Inc., 2007).

FISH was used in this research at all levels of development, for example for grid generation, input of constitutive model values, monitoring element status and forces on boundaries, and generating output files of loading, status, stress and failure histories. The modelling was conducted using modular code, in which a master file was used to call the necessary subroutines and functions. This allowed the application of the same subroutine or function to various models without modification. In order to accomplish the subroutines and functions were abstracted in such a way that they could calculate the necessary values related to model geometry and loading so that they could function. This greatly simplified the implementation of the models, although considerable effort was required to create the subroutines and functions. The following is a list of FISH and FLAC files used to run the models used in this research.
E.1.3.1 General Codes

Numerous codes for constitutive models and texture algorithms are common to several of the different tests as they were written as modules that are applicable to nearly any grid. In addition, some of the data output codes are also common to several tests as they are directly related to the arrays into which constitutive and mineralogy information is input, and were also designed as modules applicable to nearly any grid.

**Assign_mins**: several variations of this file exist, but the abstract applicability of the code is the same for all types. In some cases it was used to develop the fabric algorithms and is a simplified version of the file eventually used in testing. The file used for Brazilian tensile strength, UCS and Two Cutter testing has a copy corresponding to each of the modelled F-Factors, in which mica, quartz and feldspar content, alignment and grain size are assigned to an internal array, which is used by some of the other files, by assigning a mineral tag to each row in the array, which corresponds to an element in the model. This model also assigns two levels of variability: one at the grain level, and one at the element level.

**Batch**: several variations of the batch file exist. It is used to call several master files consecutively.

**Constitutive_array**: two variations of this code exist, but the abstract applicability of the code is the same for both types. The first was used for constitutive model development and second was a single final file used for all Brazilian, UCS and Two Cutter models for F-Factors parametric analysis. It assigns input values for the constitutive model into individual arrays for each value (i.e. cohesion, friction, etc.).

**Force**: there is a force file for each test. It performs the calculations necessary to measure the unbalanced forces and stresses at interfaces, acoustic emissions, and in some cases failed elements and strains.

**Master**: there is a master file for each test. It contains all of the subroutines and functions that are necessary to run the test, saves the file into the appropriate folder, steps the test, sets the histories and outputs all history files and status data. Some master files also output images.

**Material_assign_arrays**: assigns the input parameters contained in the arrays created by constitutive_array to individual element, such that each element has its individual input parameters depending on mineralogy and variability. It also assigns the strain softening tables created by table_assignment to FLAC specific strain softening tables for each element.
Monitor_status: monitors and updates the failure status, stress and strain at each step for each element during model stepping, and stores it in the array created by assign_mins. The strain for UCS testing is dumped into a second array at regular intervals during modelling.

Output_ms: sends the information contained in the array updated by monitor_status to a text file.

Output_strain: sends the information contained in the strain specific array created by monitor_status to a series of text files.

Table_assignment: creates tables that contain the strain-strength relationships for each input parameter. These tables are used by the model to determine peak values, as well as the strain softening behaviour.

### E.1.3.2 Fracture Toughness Test

TS_asshow: data output function that outputs array-specific data, including failure status, to the screen, which is then dumped to a text file.

TS_Force: calculates the unbalanced forces at the loading points, the deflection of the notch, failure progression away from the notch. There is a specific file for each mesh and notch geometry.

TS_geometry: creates the grid and assigns the mesh size, test dimensions and notch size.

TS_initial: applies an incremental load on the sample and assigns the fixed points at the bottom of the sample.

TS_master: master file for the fracture toughness test.

TS_material: designates the material properties for the test.

### E.1.3.3 Brazilian Tensile Strength Test

Braz_assign_mins: assigns the mineralogy, grain size and fabric to each Brazilian tensile strength test model.

Braz_constitutive_array: assigns the input properties from the constitutive model.

Braz_force: monitors the force at the top and bottom of the sample.

Braz_grid: creates the Brazilian sample grid.

Braz_initial_ramp: applies a ramped initial velocity on the platens to minimise model instability.

Braz_master: master file for the fracture Brazilian tensile strength test.
**Braz_material_assign_arrays**: assigns the input parameters and strain softening tables to each element.

**Braz_monitor_status**: monitors the status of each element at each step during modelling.

**Braz_output_ms**: outputs the information contained in the monitor status array.

**Braz_table_assignment**: creates strain softening tables for each input parameter.

### E.1.3.4 UCS Test

**UCS_assign_mins**: assigns the mineralogy, grain size and fabric to each UCS test model.

**UCS_consitutive_array**: assigns the input properties from the constitutive model.

**UCS_force**: monitors the force at the top and bottom of the sample.

**UCS_grid**: creates the UCS sample grid.

**UCS_initial_bou**: applies an initial velocity on the platens.

**UCS_master**: master file for the fracture UCS test.

**UCS_material_assign_arrays**: assigns the input parameters and strain softening tables to each element.

**UCS_monitor_status**: monitors the status of each element at each step during modelling.

**UCS_output_ms**: outputs the information contained in the monitor status array.

**UCS_output_strain**: outputs the information contained in the strain array into a series of text files.

**UCS_strain**: calculates the axial and lateral strains on the sample.

**UCS_table_assignment**: creates strain softening tables for each input parameter.

### E.1.3.5 Two Cutter Rock Interaction Model

**TC_apply_vel_cosine**: applies a velocity at the top of the cutters using a cosine function to simulate the varying force during cutter rolling.

**TC_assign_mins**: assigns the mineralogy, grain size and fabric to each Two Cutter strength test model.

**TC_consitutive_array**: assigns the input properties from the constitutive model.

**TC_cutters_small_j_l1**: assigns the steel material properties to the left cutter at the appropriate kerf depth, they have an elastic constitutive so can only elastically deform and will not fail.

**TC_cutters_small_j_r1**: same as above, for right cutter.
TC_cyclicAgain8_12_kerf_joint: subset of the master code that reapplies the cutters such that they simulate a second pass by each cutter, saving into a specified file for multiple runs, and calling the appropriate cutter files to simulate kerf depths.

TC_force_small_joint: monitors the force at the top and bottom of the sample.

TC_grid_12grid_joint_kerf: creates the small-scale Two Cutter sample grid.

TC_12grid_joint_kerf_gen: generates the location of the grid with attached joints where future kerfs will be

TC_initial_L_cutter_small_joint, R_cutter_small_joint: applies a ramped initial velocity on the cutter to minimise model instability, and fixes the sides of the cutters to prevent them moving horizontally.

TC_initial_rock: fixes the sides and bottom of the rock to simulate spacially infinite confinement.

TC_initial_rock_small_joint_stress: fixes the sides and bottom of the rock to simulate spacially infinite confinement, and applies a stress state, if using

TC_master: master file for the fracture Two Cutter test.

TC_material_assign_arrays: assigns the input parameters and strain softening tables to each element.

TC_monitor_status: monitors the status of each element at each step during modelling.

TC_output_ms: outputs the information contained in the monitor status array.

TC_output_strain: outputs the information contained in the strain array into a series of text files.

TC_table_assignment: creates strain softening tables for each input parameter.
Appendix E.2 : Numerical Model Calibration Tests

E.2.1 Introduction

This appendix contains the background details related to the development of the UCS, BTS and three-point bending beam numerical models, as well as the ellipsoid generation for rock textures and a series of results from the parametric analyses for F Factors calibration.

E.2.2 Unconfined Compression Test Development

The following section contains the geometry and boundary conditions for the UCS numerical model and a short comparison between the UCS numerical model and an analytical solution based on the same input values. The methodology and data used to determine the appropriate loading rate are presented, as well as mesh sensitivity to element size with respect to model size, and flow rule sensitivity for dilation.

E.2.2.1 UCS Test Rig in FLAC

Model parameters:

- 0.05m by 0.1m size with 100 by 200 element grid
- one element = 0.5mm
- Strain-Softening failure criterion for intact rock: friction is constant, but cohesion reduces to ¼ and tensile strength reduces to 0 after the accumulation of plastic strain (post-peak)
- Loading provided by applying an initial velocity graded throughout the sample, and fixed in time at the ends or an applied initial velocity only at top and bottom boundary. Velocity loading simulates servo controlled UCS testing.
- Velocity decreased to level where further decreases no longer affect model results.
- FISH function used to monitor unbalanced forces on the sample ends, which is turned into a stress over the area. The highest value the sample reaches is taken as the equivalent UCS value for the test.
- FISH function also monitors maximum vertical and horizontal stresses in each element, failure mode and step at which failure occurred
The derived UCS value from the FLAC model can be compared to the analytical solution developed by Jaeger and Cook (1979) and refined in the FLAC Verification Problems manual (2001), as shown in Table E.2.1. The peak stress value from the FLAC model compares very well with the analytical UCS strength. The model used for this demonstration had heterogeneous element shapes (distorted quadrilaterals) but completely homogeneous strength distribution, and for comparison, heterogeneous strength and stiffness distribution of +/- 3.25%. In this case the average strength and stiffness were the same but was normally distributed within 3.25% of the average value. The heterogeneous distribution results in a lower FLAC UCS result compared to the analytical UCS result, with respect to the percentage variation in the input strength, because the lower strength elements will yield at lower stress, and localise the failure, thereby precipitating failure of the FLAC sample at lower stress than the homogeneous model.

Analytical UCS equation for failure through intact rock:

$$\sigma_c = \frac{2c \cdot \cos \phi}{1 - \sin \phi}$$  \hspace{1cm} \text{E.2.1}$$

where $\sigma_c$ is the UCS, $C$ is cohesion of intact rock and $\phi$ is the friction angle.

### E.2.2.2 Loading Rate Sensitivity

The appropriate loading rate was determined by testing the UCS model at various loading rates and comparing their stress-strain and strain-strain curves. The highest loading rate at which no perceivable differences in stress-strain and strain-strain curves were observed was selected as the optimum loading-rate for the selected mesh size. This loading rate was selected for minimisation of impact of the loading rate, as well as minimisation of computation time. Two different approaches were tested for loading rate sensitivity: instantaneous applied velocity, and initialised graded velocity.
E.2.2.2.1 Instantaneous Applied Velocity Tests

In this configuration, the vertical velocity boundary was applied along the top and bottom model boundaries. The velocity gradient flowed through the model during computation. This configuration has the risk of shocking the boundary elements but is simple to apply, and is most representative of physical test loading conditions. Figure E.2.1 demonstrates the stress-strain and strain-strain curves for decreasing applied velocity. The velocities $1 \times 10^{-9}$ and slower show very little difference in the UCS and strain values (Figure E.2.2), and to maximise productivity, the rate of $1 \times 10^{-9}$ is used as the applied velocity for this mesh configuration (one element = 0.5mm), with a percentage difference from $5 \times 10^{-11}$ of -0.04% and -0.01% in UCS and corresponding strain, respectively (Table E.2.2).

The same exercise was performed for the UCS test with heterogeneous strength and stiffness with similar results. The differences between the faster velocities and the slowest velocity ($5 \times 10^{-10}$) were approximately 1.3% once the velocity was reduced to $2 \times 10^{-9}$. This is a higher difference than for the homogeneous material, likely arising from the variability of the material. If the same models were run several times and compared, the results would likely vary within a few percent due to the dependence of fracture initiation on the combination of stress concentration, stiffness and strength of an element, all of which will be different in different model runs.
Figure E.2.1: Stress-strain and strain-strain curves at decreasing initial applied velocity on upper and lower model boundaries. Velocity = A: $5 \times 10^{-8}$; B: $5 \times 10^{-9}$; C: $2 \times 10^{-9}$; D: $1 \times 10^{-9}$. Models with applied velocity $1 \times 10^{-9}$ and slower (D-F) result in the same peak stress, similar post peak behaviour, no pre-peak instability, and little post-peak instability. All scales are identical.
Figure E.2.1 (continued): Velocity = E: $5 \times 10^{-10}$, F: $5 \times 10^{-11}$. All scales are identical.

Figure E.2.2: Comparison graph of UCS and corresponding strain at different loading velocities, reaching asymptotic values at $1 \times 10^{-9}$ and slower.
Table E.2.2: Percentage difference values for UCS and corresponding strain at each velocity, compared with the slowest velocity, 5x10^{-11}.

<table>
<thead>
<tr>
<th>Velocity</th>
<th>Percentage UCS Difference from slowest</th>
<th>Percentage Strain Difference from slowest</th>
</tr>
</thead>
<tbody>
<tr>
<td>5e^{-8}</td>
<td>-9.56</td>
<td>16.30</td>
</tr>
<tr>
<td>5e^{-9}</td>
<td>-0.29</td>
<td>0.22</td>
</tr>
<tr>
<td>2e^{-9}</td>
<td>-0.09</td>
<td>0.07</td>
</tr>
<tr>
<td>1e^{-9}</td>
<td>-0.04</td>
<td>-0.01</td>
</tr>
<tr>
<td>5e^{-10}</td>
<td>-0.02</td>
<td>-0.01</td>
</tr>
<tr>
<td>5e^{-11}</td>
<td>0</td>
<td>0</td>
</tr>
</tbody>
</table>

E.2.2.2 Initialised Graded Velocity

In this configuration, the vertical velocity was initialised throughout the entire model by applying a velocity gradient from zero in the centre, to maximum at the top and bottom boundaries. The top and bottom velocity was fixed for the remainder of the test, while the interior velocity was allowed to change during computation. This configuration reduces the risk of shocking the boundary elements but is more complicated to apply, and is not as representative of physical test loading conditions. By reducing the risk of shocking the boundary elements, it may have been possible to increase the maximum velocity at the boundaries. However, the results in Figures E.2.3 and E.2.4 show that at velocities greater than 5x10^{-10}, the models result in higher peak and post-peak stress values and some post-peak instability. The models in this configuration (one element = 0.5mm) can be run at a maximum velocity of 1x10^{-9}, with a percentage difference of -2.67% and -1.90% for UCS and corresponding strain, respectively (Table E.2.3), showing that initialising the model with a velocity gradient does not necessarily improve the results. Note that this model includes a large inclusion with lower strength and stiffness parameters, compared with the model in Section E.2.2.2.1. The peak values are, therefore, not comparable, but the relative behaviours with respect to loading rate are.
Figure E.2.3: Stress-strain and strain-strain curves at decreasing initial velocity gradient. Boundary velocity = A: 5x10^{-8}; B: 5x10^{-9}; C: 2x10^{-9}; D: 1x10^{-9} – applied velocity at boundary 5x10^{-10}. Models with applied velocity 5x10^{-10} and slower (D-F) result in the same peak stress, similar post peak behaviour, no pre-peak instability, and little post-peak instability. All scales are identical.
Figure E.2.3 (continued): Boundary velocity = E: $5 \times 10^{-10}$, F: no gradient – applied velocity at boundary $5 \times 10^{-10}$. All scales are identical.

Figure E.2.4: Comparison graph of UCS and corresponding strain at different loading velocities, reaching asymptotic values at $5 \times 10^{-10}$ and slower.
Table E.2.3: Percentage difference values for UCS and corresponding strain at each velocity, compared with the slowest velocity, $5 \times 10^{-10}$.

<table>
<thead>
<tr>
<th>Velocity</th>
<th>Percentage UCS Difference from slowest</th>
<th>Percentage Strain Difference from slowest</th>
</tr>
</thead>
<tbody>
<tr>
<td>$5 \times 10^{-8}$</td>
<td>-47.07</td>
<td>-60.10</td>
</tr>
<tr>
<td>$5 \times 10^{-9}$</td>
<td>-5.26</td>
<td>-31.20</td>
</tr>
<tr>
<td>$2 \times 10^{-9}$</td>
<td>-5.75</td>
<td>-7.48</td>
</tr>
<tr>
<td>$1 \times 10^{-9}$</td>
<td>-2.67</td>
<td>-1.90</td>
</tr>
<tr>
<td>$5 \times 10^{-10}$</td>
<td>0</td>
<td>0</td>
</tr>
</tbody>
</table>

**E.2.2.3 Mesh Size Sensitivity**

The sensitivity of the model to element size was investigated by comparing three different model element sizes in: 50 x 100, 100 x 200 and 200 x 400 mesh sizes, with 1mm, 0.5mm and 0.25mm element sizes, respectively. The models were tested at different loading rates to remove loading rate dependencies (Figures E.2.2, E.2.5 and E.2.6), and were compared to each other at a loading rate of $5 \times 10^{-9}$, $1 \times 10^{-9}$ and $5 \times 10^{-10}$, respectively (Figure E.2.7). All other aspects of geometry and boundary conditions are the same as outlined in Section E.2.2.1. The percentage differences between the 1mm and 0.5mm element size, and between the 0.5mm and 0.25mm are below 0.05% for both UCS and corresponding strain (Table E.2.4), which is within the same order of magnitude as the percentage difference arising from velocity differences. The system, therefore, is not appreciably mesh dependent in its configuration, and the selection of mesh size is more dependent on the grain sizes being modelled rather than minimising mesh dependency.
Figure E.2.5: Comparison graph of UCS and corresponding strain at different loading velocities for the 50x100 mesh, reaching asymptotic values at $5\times10^{-9}$ and slower.

Figure E.2.6: Comparison graph of UCS and corresponding strain at different loading velocities for the 200x400 mesh, reaching asymptotic values at $5\times10^{-9}$ and slower.
Figure E.2.7: Comparison graph of asymptotic UCS values and corresponding strain for three different mesh sizes, showing very little difference in values, as long as the appropriate velocity is used.

Table E.2.4: Percentage difference values for UCS and corresponding strain for each mesh size, compared with the selected mesh size of 100x200 elements.

<table>
<thead>
<tr>
<th>Mesh Size</th>
<th>Percentage UCS Difference from 100x200</th>
<th>Percentage Strain Difference from 100x200</th>
</tr>
</thead>
<tbody>
<tr>
<td>50 x 100</td>
<td>0.04</td>
<td>0</td>
</tr>
<tr>
<td>100 x 200</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>200 x 400</td>
<td>-0.03</td>
<td>-0.26</td>
</tr>
</tbody>
</table>

E.2.2.4 Dilation Flow Rule

The sensitivity of the model to selected dilation flow rules was investigated by comparing three models with: 0, ¼ friction and ½ friction dilation. Each of the models had the same geometry and boundary conditions as outlined in Section E.2.2.1 and was loaded at an instantaneous initial velocity of \(1 \times 10^{-9}\).

Different formulations of the test were conducted, representing:

- Solid UCS
- UCS with large inclusion: hole, rubber, stiff plug, weak, and stiff-weak
These test formulations are used to investigate the interaction between two materials with contrasting material properties with respect to stiffness and peak strength. The results are used to make inferences regarding the interaction between different minerals with similarly contrasting material properties. The dilation of the minerals during yielding may or may not be important and the models were tested for sensitivity to dilation under these configurations.

For each of the test formulations the UCS and corresponding strain, post peak behaviour and yielding behaviour, in terms of location of yielded elements, were compared with respect to dilation. For all of the test formulations no difference was noted in any of the indicators investigated. These results suggest that for the configurations under investigation and homogeneous input parameters, the use of dilation between 0 and ½ friction does not affect the model results. Section 4.2.5.2.2 demonstrates the impact of 15% dilation on models with the same configuration but heterogeneous input parameters.

**E.2.3 Brazilian Test Development**

The following section contains the geometry and boundary conditions for the Brazilian test numerical model and a short comparison between the Brazilian test numerical model and an analytical solution based on the same input values. The methodology and data used to determine the appropriate loading rate are presented, as well as mesh sensitivity to element size with respect to model size, and flow rule sensitivity for dilation.

**E.2.3.1 Brazilian Test Rig in FLAC**

Model parameters:

- 0.05m diameter circle with circular 100 by 100 element grid
- one element = 0.5mm
- Strain-Softening failure criterion for intact rock: friction is constant, but cohesion reduces to ¼ and tensile strength reduces to 0 after the accumulation of plastic strain (post-peak)
- Loading provided by applying an initial velocity distributed over an arc length $2\alpha = 0.32$ at top and bottom ends of the circle instantaneously as well as incrementally applied. Velocity loading simulates servo controlled Brazilian testing.
- Velocity decreased to level where further decreases no longer affect model results.
• FISH function used to monitor unbalanced forces on the sample ends, which is turned into a stress according to equation E.2.4. The highest value the sample reaches is taken as the equivalent Brazilian tensile value for the test.

• FISH function also monitors maximum vertical and horizontal stresses in each element.

The normal and parallel stresses from the FLAC model can be compared to the analytical solution developed by Vutukuri et al. (1974) in equations E.2.2-E.2.4. The Brazilian test is only valid if failure initiates in the centre of the sample and propagates along the loading diameter, and the analytical solution assumes that the sample is homogeneous, linearly elastic and isotropic (Itasca Consulting Group Inc., 2001). Failure is assumed to be independent of stress normal to the disc face, and the numerical problem is in plane strain (Itasca Consulting Group Inc., 2001).

Analytical equations for stresses normal to the loading diameter ($\sigma_\theta$) and parallel to the loading diameter ($\sigma_r$), in the centre of the Brazilian test sample during loading:

$$\sigma_\theta = \frac{-F}{\pi r_o \alpha} \left[ \frac{\sin 2\alpha \, \left( 1 - \left( \frac{r}{r_o} \right)^2 \right)}{1 - 2 \left( \frac{r}{r_o} \right)^2 \cos 2\alpha + \left( \frac{r}{r_o} \right)^4} \right] - \frac{\tan^2 \left( \frac{\alpha \pi}{2} \right)}{2\alpha}$$  

$$\sigma_r = \frac{+F}{\pi r_o \alpha} \left[ \frac{\sin 2\alpha \, \left( 1 - \left( \frac{r}{r_o} \right)^2 \right)}{1 - 2 \left( \frac{r}{r_o} \right)^2 \cos 2\alpha + \left( \frac{r}{r_o} \right)^4} \right] + \frac{\tan^2 \left( \frac{\alpha \pi}{2} \right)}{2\alpha}$$  

where $F$ is the total applied load, $t$ is the sample thickness, $r$ is the distance from the centre, $r_o$ is the sample radius and $2\alpha$ is the arc length over which the applied force is assumed to be radially distributed (Vutukuri, Lama and Saluja, 1974), as shown in Figure E.2.8.

The normal stress at the centre of the sample, where $r = 0$, is as follows:

$$\sigma_\theta = -\frac{F}{\pi r_o t} \left[ \sin \frac{2\alpha \pi}{2} - 1 \right]$$  

This is used to calculate the sample tensile strength, provided that the failure is tensile and that failure initiates along the central diameter (Jaeger and Cook, 1979; Vutukuri, Lama and Saluja, 1974). The normal stress within the failing elements at the centre of the sample in the FLAC model are used to compare with the calculated results. The results from a distributed strip load and a line load are very similar, as long as $2\alpha$ is small and, in most applications, the Brazilian tensile strength is taken for the case where $2\alpha = 0$ (Jaeger and Cook, 1979; Vutukuri, Lama and Saluja, 1974). As shown in Figure E.2.9, the calculated tensile strength is a better
estimate for measured tensile strength when $2\alpha$ is taken into account. Results quoted in this research take $2\alpha$ into account.

As shown in Table E.2.5, the tensile (horizontal) stress value from the FLAC model is lower (~4-5%) than the analytical tensile strength calculated using the applied load. The model used for this demonstration had heterogeneous element shapes (distorted quadrilaterals) but completely homogeneous strength distribution, and for comparison, heterogeneous strength distribution +/- 3.25%. In this case the average strength was the same but was normally distributed within 3.25% of the average value. The heterogeneous strength and stiffness distribution lead to slightly lower Brazilian tensile strength results because the lower strength elements will yield at lower stress, and localise the failure, thereby precipitating failure of the FLAC sample at lower stress than the homogeneous model.
Table E.2.5: Demonstrative input values into FLAC compared to analytical solution, showing percentage difference between heterogeneous and homogeneous model strength outputs.

<table>
<thead>
<tr>
<th>Material</th>
<th>c</th>
<th>φ</th>
<th>σt</th>
<th>Calculated Tensile Strength</th>
<th>Measured Tensile Strength</th>
</tr>
</thead>
<tbody>
<tr>
<td>Homogeneous</td>
<td>40</td>
<td>9</td>
<td>10</td>
<td>10.43 MPa</td>
<td>9.99 MPa</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>9.99 MPa</td>
</tr>
<tr>
<td>Heterogeneous</td>
<td>40</td>
<td>9</td>
<td>10</td>
<td>10.51 MPa (-4.68 %)</td>
<td>9.452 MPa (-5.36 %)</td>
</tr>
<tr>
<td>+/- 3.25%</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Homogeneous</td>
<td>34</td>
<td>6.8</td>
<td>8.5</td>
<td>8.93 MPa</td>
<td>8.49 MPa</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Heterogeneous</td>
<td>34</td>
<td>6.8</td>
<td>8.5</td>
<td>8.48 MPa (-5.1 %)</td>
<td>7.90 MPa (-7.19%)</td>
</tr>
<tr>
<td>+/- 3.25%</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

**E.2.3.2 Loading Rate Sensitivity**

The appropriate loading rate was investigated by testing the Brazilian model at various loading rates and comparing their tensile strength results to the analytical solution, as well as to each other. All of the instantaneous loading rates tested did not produce a valid Brazilian tensile test since failure initiated at the platens.

In order to maximise computation time by reducing shock to the elements near the platens, the velocity was incrementally applied to the platens and the first two rows of elements were made elastic, resulting in valid model failure at all loading velocities. The calculated Brazilian tensile strengths were more sensitive to loading rate than were the measured tensile strengths, which were roughly equivalent to the input tensile strength (Table E.2.6).

A similar exercise was undertaken for the heterogeneous model discussed in Section E.2.3.1 to investigate the effect of loading rate on a model with heterogeneity. Figure E.2.10 shows the result trends are similar, in that the calculated Brazilian tensile strength is sensitive to loading rate, but trends towards the measured tensile strength (7% to 6% difference at 1x10^-9 to 5x10^-10, respectively). As shown in Table E.2.5, the measured tensile strength is lower than the average input tensile strength since it is the elements with tensile strengths from the lower end of the normal distribution that will fail first. Of importance, is the improvement in calculated results when $2\alpha$ is taken into account, because in material with heterogeneous texture, in addition to heterogeneous strength and stiffness, it is more difficult to obtain representative measured tensile strength results, and the calculated tensile strength must be shown to have enough similarity with the measured tensile strength to be depended on as an output value.
Figure E.2.9: Comparison graph of calculated (with (green) and without (blue) taking 2α into account) and measured Brazilian tensile strengths at different loading velocities. The measured values are insensitive to loading velocity, while the calculated values approach the measured values with decreasing loading velocity.

Table E.2.6: Percentage difference values for calculated and measured Brazilian tensile strength at each velocity, compared with the slowest velocity, 5x10\(^{-11}\).

<table>
<thead>
<tr>
<th>Velocity</th>
<th>Percentage Calculated Difference from slowest</th>
<th>Percentage Measured Difference from slowest</th>
</tr>
</thead>
<tbody>
<tr>
<td>5e(^{-8})</td>
<td>9.84</td>
<td>-0.01</td>
</tr>
<tr>
<td>5e(^{-9})</td>
<td>7.34</td>
<td>-1.16</td>
</tr>
<tr>
<td>2e(^{-9})</td>
<td>2.11</td>
<td>0.05</td>
</tr>
<tr>
<td>1e(^{-9})</td>
<td>4.62</td>
<td>-0.08</td>
</tr>
<tr>
<td>5e(^{-10})</td>
<td>2.56</td>
<td>0.02</td>
</tr>
<tr>
<td>5e(^{-11})</td>
<td>0</td>
<td>0</td>
</tr>
</tbody>
</table>
Figure E.2.10: Comparison graph of calculated (with (green) and without (blue) taking $2\alpha$ into account) and measured Brazilian tensile strengths at different loading velocities for the heterogeneous model. The measured values are insensitive to loading velocity, while the calculated values approach the measured values with decreasing loading velocity.

Based on the homogeneous and heterogeneous loading velocity analyses, the loading velocity should be decreased further to improve the comparability between the calculated and measured Brazilian tensile strength outputs. Computation time restrictions, however, limit the loading velocity to $1\times10^{-9}$, but both calculated and measured values are observed and used in model result interpretation.

### E.2.3.3 Mesh Size Sensitivity

The sensitivity of the model to element size was investigated by comparing three different model element sizes with: 1mm, 0.5mm and 0.25mm element sizes, respectively. The models were tested at different loading rates to remove loading rate dependencies (Figure E.2.11), and were compared to each other at a loading rate of $5\times10^{-9}$, $1\times10^{-9}$ and $5\times10^{-10}$, respectively (Figure E.2.12). All other aspects of geometry and boundary conditions are the same as outlined in Section E.2.3.1. The percentage differences between the 1mm and 0.5mm element size are below 0.2% for both calculated and measured Brazilian tensile strength (Table E.2.7), which is
within the same order of magnitude as the percentage difference arising from velocity differences. The model with element size = 0.25mm resulted in invalid Brazilian tests, regardless of loading rate, down to $1 \times 10^{-11}$, after which testing was halted due to time constraints on length of model runs. Regardless, as with the UCS model, the Brazilian model does not seem to be appreciably mesh dependent in its configuration, and the selection of mesh size is more dependent on the grain sizes being modelled rather than minimising mesh dependency.

Figure E.2.11: Comparison graph of calculated (taking $2\alpha$ into account) and measured Brazilian tensile strengths at different loading velocities, for the 50x50 element mesh. The measured values are insensitive to loading velocity, while the calculated values approach the measured values with decreasing loading velocity.
Figure E.2.12: Comparison graph of calculated and measured Brazilian tensile strength values and for two different mesh sizes, showing very little difference in values.

Table E.2.7: Percentage difference values for calculated and measured Brazilian tensile strength for each mesh size, compared with the selected mesh size of 100x100 elements.

<table>
<thead>
<tr>
<th>Mesh Size</th>
<th>Percentage Calculated Difference from 100x100</th>
<th>Percentage Measured Difference from 100x100</th>
</tr>
</thead>
<tbody>
<tr>
<td>50 x 50</td>
<td>-1.38</td>
<td>-0.81</td>
</tr>
<tr>
<td>100 x 100</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>200 x 200</td>
<td>invalid</td>
<td>invalid</td>
</tr>
</tbody>
</table>

E.2.3.4 Dilation Flow Rule

Although the main failure mode being investigated in this model is tensile, which has an associated flow rule, the sensitivity of the model to selected dilation flow rules was investigated by comparing three models with: 0, ¼ friction and ½ friction dilation for thoroughness. Each of the models had the same geometry and boundary conditions as outlined in Section E.2.3.1 and was loaded at an instantaneous initial velocity of $2 \times 10^{-9}$ and $1 \times 10^{-9}$.

For each of the test formulations the calculated and measured Brazilian tensile strength, post peak behaviour and yielding behaviour, in terms of location of yielded elements, were compared with respect to dilation. No difference was noted in any of the indicators investigated.
These results suggest that for the configurations under investigation, the use of dilation between 0 and $\frac{1}{2}$ friction does not affect the model results.

**E.2.4 Three-Point Bending Beam Test Development**

**E.2.4.1 $K_{IC}$ Test Rig in FLAC**

The three-point bending beam test was developed according to ASTM E-399 (1998) standards for mode I fracture toughness, or critical stress intensity factor $K_{IC}$, testing. The model geometry is similar to the geometry outlined in Section E.2.2.1 with element size 0.5mm, except that the dimensions are 400 x 100, width x height, and the boundary conditions are different. The boundary conditions reflect the conditions in the three-point bending beam test, as shown in Figure E.2.13.

The two bottom contact points are modelled as element nodes fixed in y, all other boundaries are free, the initial slot is a line of elements with model null constitutive model (to simulate absence of material), and the load is applied as a velocity at the nodes of five elements centred above the initial notch (Figure E.2.14). The x-displacements of the nodes at opposing side of the bottom element of the initial notch (Figure E.2.15) are summed to determine the displacement value used to interpret the $K_{IC}$ value.

![Figure E.2.13: Schematic of three-point bending beam test, modified from ASTM (1998)](image-url)
The output values for $K_{IC}$ determination are the load, $P$, and displacement, $d$. The load is calculated by summing the unbalanced $y$-forces at the nodes onto which the velocity is applied. In order to reduce system shock, the velocity is increased incrementally over $10^5$ magnitude of steps. This is necessary because, unlike the UCS test, this system is very sensitive to initial loading and minuscule velocity increments are necessary to ensure the most valid output data possible. The numerical code also monitors the load at which each element fails during testing.

Figure E.2.14: Schematic representation of boundary conditions for three-point bending beam test in a 2D FLAC model.

Figure E.2.15: Schematic representation of displacement measurement (close up of notch in Figure E.2.14)
E.2.4.2 $K_{IC}$ Determination

The $K_I$ is determined according to ASTM E-399 (1998) using the following formula:

$$K_I = \frac{3PL}{BH^{1.5}} \left[ \sqrt{\alpha \left(1.99 - \alpha (1-\alpha)(2.15 - 3.93\alpha + 2.7\alpha^2)\right)} \right]$$

where $P$, $B$, $L$ and $H$ can be found in Figure E.2.13, and $\alpha$ is determined as follows:

$$\alpha = \frac{a}{H}$$

In the 2D FLAC numerical model, $B$ is assumed to be 1.

In order to determine the appropriate load for $K_{IC}$ determination the load versus deformation graph is examined according to the methodology established in ASTM E-399 (1998), which is not reproduced here. The output load versus deformation graphs from the FLAC model are used to find the appropriate $P = \text{Load}_{\text{max}}$ according to the shape of the curve (Whittaker, Singh and Sun, 1992), and the validity tests (Equation E.2.7(ASTM, 1998)) are used to ensure the geometry of the system and the resulting $P$ and $d$ values are valid for $K_{IC}$ determination.

$$2.5\left(K_Q / \sigma_{ys}\right)^2 < B, a$$

where $K_Q$ is the $K_I$ value being tested for validity and

$$\sigma_{ys} = 1.02\sigma_t$$

and $\sigma_t$ is the tensile strength, in the case of modelling, the input tensile strength.

E.2.4.3 Loading Rate Sensitivity

Sensitivity testing was conducted on various mesh, initial and boundary condition models. For each model, the mesh was randomised and the constitutive model was strain softening, as in Section E.2.2.1, with:

- Density = 2700 kg/m$^3$
- Bulk modulus = 33e9 Pa
- Shear modulus = 20e9 Pa
- Cohesion = 20e6 Pa
- Friction = 40°
- Tensile strength = 5e6 Pa
- Dilation = 0
All failure was tensile and propagated from the notch tip toward the top of the sample. In some models, the fracture branched to the sides, but this was not very common and occurred later in the tests.

The applied velocity was tested for model sensitivity in two aspects: rate of velocity application and actual applied velocity. It was found that the model output results were more representative of laboratory curves when the velocity was applied incrementally (Figure E.2.16). Figure E.2.17 shows the load curve and load-displacement curve for the model with incremental velocity of $5 \times 10^{-11}$. The oscillations from Figure E.2.16 no longer appear, in part due to reduced actual applied velocity and increased number of incremental time steps. The pink squares in Figure E.2.17, left, are selected load at which an element failed. The first element to fail coincides with a change in slope of both curves, but the curve leading up to the failure is the linear portion of the graph used to determine the appropriate load for $K_I$ calculation. The resulting load and $K_I$ value fail the validity tests, suggesting that this geometry at this velocity is not valid. In addition, since the early portion of the load displacement curve is critical for selecting the load for $K_I$ calculation, for all tests at a velocity of $10^{-11}$ and slower, the velocity was applied incrementally over $5 \times 10^5$ steps, to further ensure a linear curve.

The test at $5 \times 10^{-11}$ is also not valid since the post-peak load increases to the point where it exceeds what should be the peak load. The curves in Figures E.2.18-E.2.20 have load-displacement curves with a linear segment leading up to the first element failure and a distinct, unique maximum load. These test velocities are considered valid tests, since the $K_I$ values calculated from the peak loads pass the validity tests. Figure E.2.21 shows the velocity dependence of the model, with seemingly asymptotic values at velocities less that $1 \times 10^{-11}$. Table E.2.8 contradicts this by showing that the $K_{IC}$ and corresponding displacement continue to decrease with decreasing applied velocity according to this relationship:

$$\Delta P \approx -2.6 \Delta velocity^{-12} \quad \text{E.2.9}$$

This relationship is related to the type of test being conducted because of the stress concentrator created by the notch (both the initial notch and the extending ‘fracture’ made up of failed elements). This relationship has also been found for bovine cortical bone (Tanabe, Tanner and Bonfield, 1998), for adhesives (Xu, Siegmund and Ramani, 2003), and for laminated composites (Hug et al, 2006).

The data plotted in Figure E.2.22 show that with decreasing velocity, the $K_{IC}$ values trend towards the $K$ value calculated using the load at which the first element failed, henceforth called $K_F$. As shown in Figure E.2.17 (left) and E.2.18 (left), the first element fails prior to the sample reaching its peak load. A pattern emerges for all models run at a velocity of $1 \times 10^{-11}$ and less: the
load-displacement curve leading up to the failure of the first element is linear with a particular slope, but the curve after this failure steepens prior to the peak load. With decreasing loading velocity, the K corresponding to the first element failure reaches a nearly asymptotic value and the failure of the first element becomes more coincident with the peak load (Table E.2.9). This suggests that the load at which the first element fails is loading rate independent, because up to that point only elastic strains have been accumulated, and it is likely that with continued decreased loading velocity, the two would coincide. This makes the \( K_F \) a suitable estimate for \( K_{IC} \), and as long as the model output \( K_{IC} \) is similar to the \( K_F \) value, then it is considered valid for the purposes of this research.

Due to the large grid size (40000 elements under this configuration) running the model at lower velocities than \( 5 \times 10^{-12} \) is prohibitively slow and since it is unclear that slower velocity will improve the results, the investigation was halted at \( 5 \times 10^{-12} \). Table E.2.9 shows that at a velocity of \( 9 \times 10^{12} \) the difference between \( K_{IC} \) and \( K_F \) is \( \approx 15\% \), while at \( 7 \times 10^{-12} \) it is \( \approx 9\% \). For this reason, many of the tests are run at both velocities, to ensure that the results are not compromised by the loading velocity.

![Figure E.2.16: Comparison of load curves at instantaneous (left) and incremental over 7x10^3 time steps (right) velocity of 5x10^{-10} demonstrating the difference in the magnitude of oscillation of the load curve. Note the reduction in oscillation magnitude with incremental velocity application. The velocity is still too high, however for these results to be valid.](image)
Figure E.2.17: Left: load curve for incremental (7x10^5) velocity 5x10^-11 showing load at which selected elements failed (pink squares); Right: load-displacement curve for incremental velocity 5x10^-11 showing OA and OPq lines used to determine the load, P, for K, calculation according to ASTM (1998).

Figure E.2.18: Left: load curve for incremental (5x10^5) velocity 1x10^-11 showing load at which selected elements failed (pink squares); Right: load-displacement curve for incremental velocity 1x10^-11
Figure E.2.19: Left: load-displacement curve for incremental velocity $9\times10^{-12}$; Right: load-displacement curve for incremental velocity $7\times10^{-12}$

Figure E.2.20: load-displacement curve for incremental velocity $5\times10^{-12}$
Figure E.2.21: Comparison graph of $K_{IC}$ and corresponding displacement at different loading velocities, never reaching asymptotic values.

Table E.2.8: Percentage difference values for load and displacement at each velocity, compared with the slowest velocity, $5 \times 10^{-12}$.

<table>
<thead>
<tr>
<th>Velocity</th>
<th>Percentage Load Difference from slowest</th>
<th>Percentage Displacement Difference from slowest</th>
</tr>
</thead>
<tbody>
<tr>
<td>$5 \times 10^{-11}$</td>
<td>-101.89</td>
<td>-178.62</td>
</tr>
<tr>
<td>$1 \times 10^{-11}$</td>
<td>-12.94</td>
<td>-3.09</td>
</tr>
<tr>
<td>$9 \times 10^{-12}$</td>
<td>-10.87</td>
<td>-2.93</td>
</tr>
<tr>
<td>$7 \times 10^{-12}$</td>
<td>-4.94</td>
<td>-2.68</td>
</tr>
<tr>
<td>$5 \times 10^{-12}$</td>
<td>0</td>
<td>0</td>
</tr>
</tbody>
</table>
Figure E.2.22: Comparison graph of $K_{IC}$ and $K_F$ at different loading velocities; at the values trend toward each other with decreasing applied loading velocity.

Table E.2.9: Percentage difference values $K_{IC}$, compared with $K_F$ at each velocity.

<table>
<thead>
<tr>
<th>Velocity</th>
<th>Percentage $K_{IC}$ Difference from slowest</th>
<th>Percentage $K_F$ Difference from slowest</th>
<th>Percentage $K_{IC}$ Difference from $K_F$</th>
</tr>
</thead>
<tbody>
<tr>
<td>$1 \times 10^{-11}$</td>
<td>7.55</td>
<td>8.73</td>
<td>-112.03</td>
</tr>
<tr>
<td>$1 \times 10^{-11}$</td>
<td>-2.01</td>
<td>0</td>
<td>-14.79</td>
</tr>
<tr>
<td>$9 \times 10^{-12}$</td>
<td>-1.38</td>
<td>0</td>
<td>-14.99</td>
</tr>
<tr>
<td>$7 \times 10^{-12}$</td>
<td>-0.83</td>
<td>0</td>
<td>-8.68</td>
</tr>
<tr>
<td>$5 \times 10^{-12}$</td>
<td>0</td>
<td>0</td>
<td>-4.75</td>
</tr>
</tbody>
</table>

**E.2.4.4 Mesh and Notch Size Sensitivity**

The model required testing for geometry and boundary condition sensitivity. The mesh sensitivity testing discussed in Section E.2.2.3 demonstrated that the model was not sensitive to mesh ranging from 1.5mm-0.5mm element size. The geometry in this test is similar, but the boundary conditions, and in particular, the stress concentrator at the notch tip make this model more sensitive to mesh size. The mesh size was tested with a 200x50 and a 75x300 mesh to compare with the 100x400 mesh from Section E.2.4.3.

The same trend of decreasing peak load and $K_{IC}$ with decreasing applied velocity for the two additional tested mesh sizes leads to the absence of an asymptotic or average or
representative value for $K_{IC}$ that could be used for each mesh configuration to compare to the 100x400 mesh. The percentage difference between models could vary greatly depending on the velocity at which the results were taken for comparison. To minimise this uncertainty, the velocity (in m/step) was converted to strain rate per step according to the element size, and all three mesh sizes were compared at the same elemental strain rate. The results in Table E.2.10 show that the model is very mesh dependent, likely due to increased localisation caused by the interaction of the stress concentrator at the notch tip with decreasing element size. This is contrary to grain size dependence results discussed in Whitakker et al. (1992), in which the fracture toughness increases with decreased grain size. For this reason, element size cannot be used as a proxy for grain size in this homogeneous mono-mineralic system.

This mesh dependency holds for comparison between $K_{IC}$ values calculated according to ASTM E-339 (1998) and using the load at first element failure, $K_F$, suggesting that the sensitivity arises from the deformation/strain/stress relationship in the model (Figure E.2.23). It should be noted that the load at first element failure in the model with 50 x 200 elements was sensitive to loading rate, suggesting that the element size is too large for the results to be meaningful for this particular test. The $K_F$ results in mesh size 75 x 300 were found to be insensitive to loading rate, as was demonstrated for the 100 x 400 mesh in Table E.2.9.

Table E.2.10: Percentage difference values for $K_{IC}$ and displacement at $1 \times 10^{-8}$ elemental strain rate for three different mesh sizes.

<table>
<thead>
<tr>
<th>Mesh Size</th>
<th>$K_{IC}$ (MPa$m^{0.5}$)</th>
<th>Displacement</th>
<th>$% K_{IC}$ Difference</th>
<th>$%$ Displacement Difference</th>
</tr>
</thead>
<tbody>
<tr>
<td>50 x 200</td>
<td>0.26</td>
<td>1.52 x $10^{-6}$</td>
<td>67</td>
<td>23</td>
</tr>
<tr>
<td>75 x 300</td>
<td>0.19</td>
<td>1.36 x $10^{-6}$</td>
<td>37</td>
<td>12</td>
</tr>
<tr>
<td>100 x 400</td>
<td>0.13</td>
<td>1.23 x $10^{-6}$</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
Figure E.2.23: Schematic of initial notch width impact on load required to extend fracture. The crack extension is a function of the load and the width (dashed curves). An infinitesimally narrow initial notch would result in an infinitesimally high stress concentration at the notch tip, and an infinitesimally long crack extension (dash-dot lines). Progressively wider initial notches would result in progressively lower stress concentrations at the tip, and the crack extension would be lower for a given load. For a crack to extend one element ahead of the notch, the applied load must be greater for a wider initial notch, and similarly for larger element sizes.

Although it was shown that the model is mesh sensitive, and models with element sizes less than 0.5mm were not tested due to impracticable model sizes and run times, the model with 100 x 400 elements was selected for tensile strength testing because it reflects the mesh size under consideration for this investigation. The problem under investigation involves the interactions at the element scale, which are 0.5mm in length, and the most representative calibration model to obtain strength inputs for the investigation has the same element size.

In addition to mesh size sensitivity, the effect of notch width was also investigated. This was investigated in the 100x400 mesh configuration at the same applied velocity, for a notch width of 2 and 3 elements compared with the original notch width of 1 element. The results in Table E.2.11 demonstrate the sensitivity of the model to notch width, as shown in Figure E.2.23. This has also been observed in alumina by Munz et al. (1980) and Mussler et al (1982). Note that the differences in $K_{ic}$ arising from notch width are also loading velocity dependent. The results in Table E.2.11 were loaded at a velocity of $9 \times 10^{-12}$, but the same test loaded at a velocity of $7 \times 10^{-12}$ results in 25-30% lower differences between models with differing notch widths.
Table E.2.11: Percentage difference values for $K_{IC}$ and displacement for different initial notch widths.

<table>
<thead>
<tr>
<th>Notch Width (mm)</th>
<th>$K_{IC}$ (MPam$^{0.5}$)</th>
<th>Displacement</th>
<th>% $K_{IC}$ Difference</th>
<th>% Displacement Difference</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.5</td>
<td>0.14</td>
<td>1.25 x 10^{-6}</td>
<td></td>
<td></td>
</tr>
<tr>
<td>1</td>
<td>0.17</td>
<td>1.43 x 10^{-6}</td>
<td>15.12</td>
<td>13.43</td>
</tr>
<tr>
<td>1.5</td>
<td>0.18</td>
<td>1.53 x 10^{-6}</td>
<td>20.61</td>
<td>20.14</td>
</tr>
</tbody>
</table>

The selection of the appropriate notch width depends on the goal of the testing. If the goal is to precisely simulate a three-point bending beam test, then the notch width should be the same as the width prescribed in testing specifications, such as ASTM E-399 (1998). If the goal is to identify the relationship between input values and output results, such as input tensile strength and output fracture toughness, for particular configurations, then the notch size should reflect the problem. In this investigation, the yielding mechanism of interest is on an element basis (as described in Section 4.2.3.2), and the stress concentrator will also be at the element scale, so the notch width of 1 element is most applicable to the problem being investigated.

The significant digits used in the previous data have no importance, but at least two decimal places were used to allow for differentiation between the results from different models. In addition, for the $K_{IC}$ values to be meaningful, at least two significant digits are necessary. This does not reflect any error propagation, nor is it related to confidence in the precision of the test. FLAC is a double precision code and carries 16 decimal places in scientific notation, but this level of precision can be misleading and should not be quoted in results due to errors in model creation, input parameters, initial and boundary conditions, interpretation of data, etc.

The model required testing for geometry and boundary condition sensitivity. The mesh sensitivity testing discussed in Section E.2.4.3 demonstrated that the model was not sensitive to mesh ranging from 1mm-0.25mm element size. The geometry in this test is similar, but the boundary conditions, and in particular, the stress concentrator at the notch tip make this model more sensitive to mesh size. The mesh size was tested with a 200x50 and a 75x300 mesh to compare with the 100x400 mesh from Section E.2.4.3.
Appendix E.3 : Simulation of Realistic Rock Textures

E.3.1 Introduction

As stated in Section 5.1, the goal of the numerical modelling of simple geometries is to calibrate the relative impacts different textures have on fracture initiation and propagation, and resulting spall sensitivity. In order to accomplish this, a constitutive model for each mineral type was first developed (Section 5.2). The second aspect to employing mineral-specific properties to the numerical model is to apply the constitutive models to mineral grain analogues. The finite difference mesh in FLAC is made up of elements, each of which can be assigned different constitutive model properties, making it possible to simulate mineral grains. From this, the impact of neighbouring grains on each other in terms of initiators and receivers of tensile fractures can be investigated as a function of strength/stiffness/dilation properties, geometry and alignment with respect to the induced stress.

Three alternatives were considered for generating mineral grains, each with increasing implementation complexity. The first alternative is the random assignment of mineral properties to each element within a numerical model. The resulting aggregates of equivalent constitutive properties can be considered as mineral grains. This alternative is simple, but the grain size is dependent on random aggregates forming and is a function of the element size. The second alternative is the duplication of real rock textures from digital raster photomicrographs by assigning mineral properties as seen in the photo to x,y coordinates in the numerical model (Chen et al, 2007; Li et al, 2003). This alternative generates very realistic textures, but it must be simplified according to element size and photo resolution and does not allow for control on grain size or alignment for parametric analysis.

The third alternative is the generation of mineral aggregates using simple geometrical algorithms for generating ellipsoids. This alternative allows explicit control on aggregate size and orientation through specification of ellipsoid size and elongate axis orientation. In addition, depending on element size versus modelled grain size, grain boundaries can also be explicitly generated. This alternative is preferentially used in the Geomechanical Characterisation scheme calibration.
**E.3.2 Equations for Ellipsoid generation**

The basic equation for finding the distance, $r$, from one focus of an ellipse to a point on the edge of the ellipse at angle $\theta$ from the long axis, $\ast$, is (Figure E.3.1):

\[ r = \frac{ed}{1 + e\cos \theta} \]  

E.3.1

where,

\[ d = \frac{c(1-e^2)}{e^2} \]  

E.3.2

and,

\[ c = ea \]  

E.3.3

Subbing equations E.3.2 and E.3.3 into E.3.4, it can be simplified to only three variables:

\[ r = \frac{a(1-e^2)}{1 + e\cos \theta} \]  

E.3.4

For the purposes of this research the angle of the long axis with respect to horizontal, $\gamma$, must have the capability to be varied (Figure E.3.2). In order to accommodate this, the basic formula for determining $r$ was modified by first determining the necessary angles (Figure E.3.3). First, the $\gamma$ angle is prescribed or randomly generated within the FISH code (programming language internal to FLAC, see Appendix E.1). Second, the $\alpha$ angle relating the line between the focus and the test element to the horizontal is calculated as:

\[ \alpha = \tan^{-1} \left( \frac{y_{testelement} - y_{focus}}{x_{testelement} - x_{focus}} \right) \]  

E.3.5

Thirdly, the $\theta$ angle between the long axis and the line between the focus and the test element is calculated as:

\[ \theta = 180 - \alpha - \gamma \]  

E.3.6
Figure E.3.1: Schematic ellipse showing the distance relations between one of the foci and the edge of the ellipse.

Once $\theta$ has been determined, it can be used to solve equation E.3.4 to obtain the distance between the focus and the ellipsoid boundary along the line between the focus and the test element, $r$. The distance $t$ is then tested against the distance $r$ to determine whether or not the test element is inside or outside the ellipsoid.

The FISH code contains a series of ‘if’ statements to allow the code to determine the lengths of $t$ and $r$ for test elements within any quadrant of the x-y coordinate plane, and for any orientation of the long axis of the ellipsoid. The ellipsoid dimensions are prescribed in the FISH code as the half-length of the long axis, $a$, and of the short axis, $b$. A circle is generated by making $a=b$. A second set of ‘if’ statements assigns grain boundary properties to the elements within one element dimension from the ellipsoid boundary according to the mineral type being assigned and the mineral type adjacent to it.
Figure E.3.2: schematic of ellipse rotated g degrees from horizontal

Figure E.3.3: schematic of rotated ellipse section showing necessary angles for determining r for a test element.
E.3.3 Algorithm for Generating Aggregates

E.3.3.1 Mineral Shapes

Natural crystal and grain geometries were used as the model for generating mineral shapes. A quick overview showed that most minerals can either be represented as circles or ellipses (Figure E.3.4). For this reason, ellipsoids were selected for the mineral aggregate generation algorithm (circles being ellipsoids with no eccentricity). By changing the eccentricity and radius of ellipsoids, various mineral shapes could be generated (Figure E.3.5). In addition, the orientation of the elongate axis (Figure E.3.6) can be specified to generate foliation textures. The shapes are kept simple to reduce complexity and uncertainty in the model. All textures generated by the ellipsoid algorithm are simplifications of real rock textures to highlight the interactions between mineral grains in a clearly definable way.

Figure E.3.4: Photomicrographs of round and elliptical grain shapes; clockwise from top left: feldspar, feldspar, quartz, mica and feldspar.
Figure E.3.5: Examples of circular (left) and ellipse (right) grain shapes.
As discussed in Section 5.2.3.6, mineral grains have grain boundaries, which are either a misalignment of two equivalent mineral lattices or boundaries between dissimilar mineral types. Regardless of grain boundary type, their presence can affect fracture processes by acting either as fracture initiators (Brace et al, 1972; Diederichs, 1999; Moore and Lockner, 1995; Tapponier and Brace, 1976) or attenuators below critical damage due to heterogeneous force distributions (Diederichs, 1999; Martin, 1994). To include these effects in the numerical model, grain boundaries are simulated with the grain generation algorithm (Figure E.3.7).
E.3.3.2 Mineral Properties

Once the mineral shapes are generated by assigning each element in the numerical model a mineral status in a FISH array (Itasca’s built-in programming language), the constitutive model appropriate to each individual mineral type, as discussed in Section 5.2, is assigned. The grain boundaries are assigned in a similar way, but their constitutive models are modifications of the mineral-specific constitutive models determined in Section 5.2.

The misalignment of mineral lattices increases spacing between molecules making up mineral grains, allows the inclusion of impurities and minuscule mineral fragments, pores, and cracks between grains and reduces the tension and cohesion between different grains (Diederichs, 1999). All of these contribute to a decrease in stiffness and a decrease in strength at the grain boundary. In order to reflect these property changes in the constitutive model, the stiffness and strength parameters are reduced. The selection of grain boundary parameters is related to the two adjacent mineral grains making up the boundary, as discussed in Section 5.2.3.6.

E.3.4 Algorithm for Generating Texture

E.3.4.1 Rock Textures

Rock textures simulated in the numerical model are comprised of the mineral modal percentages, the grain geometries and their orientations, and are based on samples collected from...
tunnelling projects in the Swiss Alps. The textures to be simulated are selected for their geomechanical characterisation, that is, their F Factors. One representative sample was selected for each combination of F Factors for use in quantification of the relative impact each factor has on fracture processes. The basic building blocks for texture simulation are the ellipsoid algorithm for generating mineral aggregates and the mineral-specific constitutive models for minerals and their grain boundaries.

The final tool for generating structure is a hierarchy of governing mineral relationships built into a FISH function that overlays different mineral types to obtain the appropriate texture. These rules are roughly based on observation of mineral grain relationships for each rock texture. For example, in Figure E.3.8, the relationships between mica, quartz and feldspar grains are as follows: the feldspar grains are either elongate porphyroclasts or ragged-edged grains; the quartz grains are interlocking as either individual rounded grains or elongate quartz ribbons with considerable subgrain development; the micas are very thin, elongate grains defining a distinct lineation (in this 2-D image) parallel to the elongate feldspars and quartz ribbons. In order to simulate this texture a set of algorithms in which certain mineral types and shapes are generated in a particular order with overprinting rules is necessary to create the mineral grain types and their interactions. These overprinting rules are texture-specific rules used during the ellipsoid generation algorithm, specifying whether or not an element, which was already assigned a mineral status, should remain the same or be assigned the new status.

### E.3.4.2 Generating Mineral Modal Percentages

Generating a representative mineralogy is the first criteria for validation of the simulated rock texture. Each element in the model is assigned a mineral-specific constitutive model by means of a status designation in a FISH array. During status assignment the texture generating algorithm monitors the number of elements that have been assigned a particular mineral status. A maximum value is input into the algorithm to ensure that it ceases to assign a particular mineral status once this value has been reached. The maximum value will differ depending on the modal percentages to be modelled as well as the texture generation overprint rules and is determined through an iterative process. The modal percentage of the generated texture is recorded in the FISH array and is used to adjust the maximum values.
In most cases, considerably more elements must be converted to a particular mineral status because a large percentage of these elements will be overprinted by the ellipsoid generating algorithm. The greater the number of times the algorithm is applied (in generating different mineral types, geometries or orientations) the larger the maximum number must be, in particular in the earlier applications of the algorithm. The overprint rules will similarly require adjusted maximum numbers, such that minerals that may be overprinted will require higher numbers than those that may not be overprinted.

**E.3.4.3 Generating Grain Size and Grain Size Distribution**

Generating grain size and grain size distribution are the second criteria required for simulation of texture. The grain size is specified during ellipsoid generation by the radius or height and width of the ellipsoid. The distribution of grain size is specified by generating different ellipsoid sizes for equivalent mineral types. The overprinting rules will affect the grain
size and distribution outcome by modifying the original grain sizes specified during early applications of the ellipsoid algorithm. For example, an elongate ellipse generated early will not necessarily retain this shape if the mineral aggregate is overprinted by subsequent applications of the ellipsoid algorithm (Figure E.3.9).

The order in which the ellipsoid algorithm for different mineral types is applied will determine the outcome of the grain shapes and sizes, which will affect the distribution. The grains with the smallest size and random shape, and in particular random spatial distribution (not size distribution) are generated first, in some cases assigned to each element in the model without the ellipsoid algorithm. These small grains will be ‘left over’ as isolated elements or very small aggregates once the ellipsoid algorithms have been applied. The grains with the most distinct shapes or largest grain sizes are generated last so as not to be overprinted by subsequent applications of the ellipsoid algorithm. Finally a balance between grain size and the mineral type’s role in defining a foliation will govern the order in which the ellipsoid algorithms are applied, the original grain size specified and the number of applications of the algorithm.

Figure E.3.9: Images of ellipsoids (yellow with pink boundaries) overlapping each other.
E.3.4.4 Generating Foliation with Elongate Mineral Orientation

The ellipsoid algorithm contains a segment in which the orientation of the elongate axis of an ellipse can be specified. The orientation can be randomly generated, for textures without foliation, aligned in one particular direction, any number of times to create any number of foliations and lineations, and the orientation itself can be randomly selected from a range, to provide some heterogeneity in the alignment.

As discussed in previous subsections, the overprint rules in subsequent applications of the ellipsoid algorithm for different minerals can impact the resultant texture. For example, aligned ellipses can lose their impact in defining a foliation if they are consistently overprinted during texture generation (Figure E.3.10). The overprint rules must reflect the end product in such a way that minerals whose shapes, orientations and boundaries define a clear fabric should not be overprinted in the generation of subsequent mineral aggregates.

Figure E.3.10: Image of ellipsoids (in red, left and in blue, right) overlapping previously generated ellipsoids (in blue, left, and in red and yellow, right).

E.3.4.5 Generated Textures

An iterative process was used to determine the order in which mineral aggregates were generated using the ellipsoid algorithm, the number of times the algorithm was applied for each mineral type and the rules governing overprinting of a pre-existing mineral status in an element by a subsequent application of the ellipsoid algorithm. The initial trials were based on examination of mineralogy, desired grain size and grain size distribution, the mineral shapes and orientations, which minerals define a fabric, the relationships between different mineral types and
grain boundary geometries. Representative samples collected in Swiss tunnelling projects that illustrate each combination of F-factors were used. Figures E.3.11 – E.3.13 are illustrations of sample photomicrographs and their corresponding generated textures.

Figure E.3.11: Photomicrograph of sample GA_a072 and image of modelled domainal schistosity fabric.

Figure E.3.12: Photomicrograph of sample GA_c127 and image of modelled type 1 schistosity fabric.
Figure E.3.13: Photomicrograph of sample GA_a091 and image of modelled cleavage fabric with 0.5mm-5mm spacing.

Mineralogy

- **Feldspar**
- **Grain Boundaries**
- **Quartz**
- **Biotite**

Figure E.3.14: Legend for Figures E.3.11 – E.3.13.
8.14 Appendix 5.3: F Factors Calibration Results

8.14.1 Introduction

The Appendix contains the data resulting from the parametric analysis modelling using the Brazilian tensile strength and UCS numerical model developed in Appendix D.2 to calibrate the F Factors. A total of 57 individual test configurations were used and for each one a two-page layout of the test results are presented in this appendix. The first page of each sample layout contains an introduction to the sample and images of the failed elements at 75%, 95% and peak strength, strain patterns at 95% and peak strength and an image of the mineralogy distribution. All figures are the same, and the legend for all of them is in Figure E.4.1. The strain figures have no common legend, however strain is highest at locations of lighter colour in contrast to background burgundy-red, and increase with cooler colours. The second page contains a table describing the input parameters and strength and failure process output values as well as a graph with stress-strain and stress-acoustic emission curves. This appendix can be found in the CD appended to the thesis.

<table>
<thead>
<tr>
<th>Mineralogy</th>
<th>Failure</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Previous Failure, no longer active</td>
</tr>
<tr>
<td></td>
<td>Active Tensile Failure</td>
</tr>
<tr>
<td></td>
<td>Active Shear Failure</td>
</tr>
<tr>
<td></td>
<td>Previous Failure along ubiquitous joint, no longer active</td>
</tr>
<tr>
<td></td>
<td>Active Tensile Failure along ubiquitous joint</td>
</tr>
<tr>
<td></td>
<td>Active Shear Failure along ubiquitous joint</td>
</tr>
</tbody>
</table>

Figure E.4.1: Legend for Figures 8.14.2-8.14.115.
**E.4.1 Brazilian Indirect Tensile Strength Mica Variation 1 Sample**

The goal of the mica variation 1 (M1) sample was to increase the mica content, while keeping the quartz to feldspar ratio the same.

![Failure maps and mineralogy images](image)

Figure E.4.2: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

| **Table E.4.1: Input and output parameters from sample M1.** |
|-----------------|-----------------|
| **Strength Indicators** | **Mineralogy** |
| Brazilian Indirect Tensile (MPa) | Content |
| 17.3 | Mica | 6 |
| | Quartz | 20 |
| | Feldspar | 74 |
| | Q:F ratio | 0.27 |
E.4.2 Brazilian Indirect Tensile Strength Mica Variation 2 Sample

The goal of the mica variation (M2) sample was to increase the mica content, while keeping the quartz to feldspar ratio the same.

Figure E.4.3: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

Table E.4.2: Input and output parameters from sample M2.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>Content</td>
</tr>
<tr>
<td>15.3</td>
<td></td>
</tr>
<tr>
<td>Mica</td>
<td>18</td>
</tr>
<tr>
<td>Quartz</td>
<td>17</td>
</tr>
<tr>
<td>Feldspar</td>
<td>65</td>
</tr>
<tr>
<td>Q:F ratio</td>
<td>0.26</td>
</tr>
</tbody>
</table>
E.4.3 Brazilian Indirect Tensile Strength Mica Variation 3 Sample

The goal of the mica variation (M3) sample was to increase the mica content, while keeping the quartz to feldspar ratio the same.

Table E.4.3: Input and output parameters from sample M3.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect</td>
<td>Content</td>
</tr>
<tr>
<td>Tensile (MPa)</td>
<td></td>
</tr>
<tr>
<td>14.5</td>
<td></td>
</tr>
<tr>
<td>Mica</td>
<td>25</td>
</tr>
<tr>
<td>Quartz</td>
<td>17</td>
</tr>
<tr>
<td>Feldspar</td>
<td>58</td>
</tr>
<tr>
<td>Q:F ratio</td>
<td>0.29</td>
</tr>
</tbody>
</table>
E.4.4 Brazilian Indirect Tensile Strength Quartz Variation 1 Sample

The goal of the quartz variation (Q1) sample was to increase the quartz content, while keeping the mica to feldspar ratio the same.

Figure E.4.5: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

Table E.4.4: Input and output parameters from sample Q1.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
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<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>Mica</td>
<td>Content</td>
</tr>
<tr>
<td>14.8</td>
<td>Quartz</td>
<td>10</td>
</tr>
<tr>
<td></td>
<td>Feldspar</td>
<td>41</td>
</tr>
<tr>
<td></td>
<td>Q:F ratio</td>
<td>0.84</td>
</tr>
</tbody>
</table>
The goal of the quartz variation (Q2) sample was to increase the quartz content, while keeping the mica to feldspar ratio the same.

Figure E.4.6: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

Table E.4.5: Input and output parameters from sample Q2.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
<th>Content</th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>Mica</td>
<td>6</td>
</tr>
<tr>
<td></td>
<td>Quartz</td>
<td>70</td>
</tr>
<tr>
<td></td>
<td>Feldspar</td>
<td>24</td>
</tr>
<tr>
<td></td>
<td>Q:F ratio</td>
<td>2.92</td>
</tr>
</tbody>
</table>
E.4.6  **Brazilian Indirect Tensile Strength Quartz Variation 3 Sample**

The goal of the quartz variation (Q3) sample was to increase the quartz content, while keeping the mica to feldspar ratio the same.

![Image of failure maps and mineralogy](image)

Figure E.4.7: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
<th>Content</th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>20.3</td>
<td></td>
</tr>
<tr>
<td>Mica</td>
<td>3</td>
<td></td>
</tr>
<tr>
<td>Quartz</td>
<td>84</td>
<td></td>
</tr>
<tr>
<td>Feldspar</td>
<td>13</td>
<td></td>
</tr>
<tr>
<td>Q:F ratio</td>
<td>6.46</td>
<td></td>
</tr>
</tbody>
</table>

Table E.4.6: Input and output parameters from sample Q3.
E.4.8  **Brazilian Indirect Tensile Strength Quartz Variation 4 sample**

The goal of the quartz variation 4 (Q4) sample was to increase the quartz to feldspar ratio, while keeping the mica content the same.

![Figure E.4.9: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.](image)

**Table E.4.8: Input and output parameters from sample Q4.**

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>Content</td>
</tr>
<tr>
<td>16.6</td>
<td></td>
</tr>
<tr>
<td>Mica</td>
<td>12</td>
</tr>
<tr>
<td>Quartz</td>
<td>17</td>
</tr>
<tr>
<td>Feldspar</td>
<td>71</td>
</tr>
<tr>
<td>Q:F ratio</td>
<td>0.24</td>
</tr>
</tbody>
</table>
**E.4.9 Brazilian Indirect Tensile Strength Quartz Variation 5 Sample**

The goal of the quartz variation (Q5) sample was to increase the quartz to feldspar ratio, while keeping the mica content the same.

![Figure E.4.10: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.](image)

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
<th>Content</th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>15.9</td>
<td>Mica</td>
<td>11</td>
</tr>
<tr>
<td></td>
<td>Quartz</td>
<td>43</td>
</tr>
<tr>
<td></td>
<td>Feldspar</td>
<td>46</td>
</tr>
<tr>
<td></td>
<td>Q:F ratio</td>
<td>0.93</td>
</tr>
</tbody>
</table>
**E.4.10 Brazilian Indirect Tensile Strength Quartz Variation 6 sample**

The goal of the quartz variation (Q6) sample was to increase the quartz to feldspar ratio, while keeping the mica content the same.

Figure E.4.11: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

Table E.4.10: Input and output parameters from sample Q6.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>Content</td>
</tr>
<tr>
<td>16.9</td>
<td></td>
</tr>
<tr>
<td>Mica</td>
<td>12</td>
</tr>
<tr>
<td>Quartz</td>
<td>70</td>
</tr>
<tr>
<td>Feldspar</td>
<td>18</td>
</tr>
<tr>
<td>Q:F ratio</td>
<td>3.89</td>
</tr>
</tbody>
</table>
**E.4.12 Brazilian Indirect Tensile Strength Quartz Variation 7 Sample**

The goal of the quartz variation (Q7) sample was to increase the quartz to feldspar ratio, while keeping the mica content the same.

![Image](image_url)

Figure E.4.13: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

Table E.4.12: Input and output parameters from sample Q7.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>Content</td>
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<tr>
<td>17.5</td>
<td>Mica</td>
</tr>
<tr>
<td></td>
<td>Quartz</td>
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<td></td>
<td>Feldspar</td>
</tr>
<tr>
<td></td>
<td>Q:F ratio</td>
</tr>
</tbody>
</table>
E.4.7  
**Brazilian Indirect Tensile Strength Quartz Variation 35 Sample**

The goal of the quartz variation (Q35) sample was to increase the quartz content, while keeping the mica content very low.

![Failure maps and mineralogy images](image)

Figure E.4.8: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

Table E.4.7: Input and output parameters from sample Q35.

<table>
<thead>
<tr>
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<th><strong>Mineralogy</strong></th>
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</thead>
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<td>Mica</td>
<td>4</td>
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<tr>
<td>Quartz</td>
<td>91</td>
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<tr>
<td>Feldspar</td>
<td>5</td>
</tr>
<tr>
<td>Q:F ratio</td>
<td>18.2</td>
</tr>
</tbody>
</table>
E.4.11  Brazilian Indirect Tensile Strength Quartz Variation 65 sample

The goal of the quartz variation (Q65) sample was to increase the quartz to feldspar ratio, while keeping the mica content the same.

![Failure map and strain map](image)

Figure E.4.12: From top-right: failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

Table E.4.11: Input and output parameters from sample Q65.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
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<th>Content</th>
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<td>Brazilian Indirect Tensile (MPa)</td>
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</tr>
<tr>
<td>17.7</td>
<td>Quartz</td>
<td>80</td>
</tr>
<tr>
<td></td>
<td>Feldspar</td>
<td>8</td>
</tr>
<tr>
<td></td>
<td>Q:F ratio</td>
<td>10</td>
</tr>
</tbody>
</table>
The goal of the grain size 2 (G2) sample was to generate a sample with an average grain size of 4mm with grain boundaries one element thick.

Figure E.4.18: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

Table E.4.17: Input and output parameters from sample G2.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>14.3</td>
</tr>
<tr>
<td>Mica</td>
<td>9</td>
</tr>
<tr>
<td>Quartz</td>
<td>25</td>
</tr>
<tr>
<td>Feldspar</td>
<td>66</td>
</tr>
<tr>
<td>Q:F ratio</td>
<td>0.38</td>
</tr>
</tbody>
</table>
**E.4.18 UCS Grain Size 3 (8mm with grain boundaries)**

**Sample**

The goal of the grain size 3 (G3) sample was generate a sample with an average grain size of 8mm with grain boundaries one element thick.

![Figure E.4.19: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.](image)

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
<th>Content</th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>Mica</td>
<td>8</td>
</tr>
<tr>
<td></td>
<td>Quartz</td>
<td>17</td>
</tr>
<tr>
<td></td>
<td>Feldspar</td>
<td>75</td>
</tr>
<tr>
<td></td>
<td>Q:F ratio</td>
<td>0.23</td>
</tr>
</tbody>
</table>
E.4.19  **Brazilian Indirect Tensile Strength Grain Size 4 (12mm with grain boundaries) Sample**

The goal of the grain size 4 (G4) sample was generate a sample with an average grain size of 12mm with grain boundaries one element thick.

![Figure E.4.20: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.](image)

Table E.4.19: Input and output parameters from sample G4.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
<th>Content</th>
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</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
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<tr>
<td>14.4</td>
<td>Mica</td>
<td>6</td>
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<tr>
<td></td>
<td>Quartz</td>
<td>23</td>
</tr>
<tr>
<td></td>
<td>Feldspar</td>
<td>71</td>
</tr>
<tr>
<td></td>
<td>Q:F ratio</td>
<td>0.32</td>
</tr>
</tbody>
</table>
E.4.21  **Brazilian Indirect Tensile Strength Domainal Schistosity Sample**

The goal of the domainal schistosity (DS) sample was to generate a fabric in which the mica grains are aligned, continuously connected across the sample, visible to the naked eye and anastamosing around feldspar and quartz microlithons.

![Figure E.4.22: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.](image)

<table>
<thead>
<tr>
<th><strong>Strength Indicators</strong></th>
<th><strong>Mineralogy</strong></th>
<th><strong>Content</strong></th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>Mica</td>
<td>13</td>
</tr>
<tr>
<td></td>
<td>Quartz</td>
<td>24</td>
</tr>
<tr>
<td></td>
<td>Feldspar</td>
<td>63</td>
</tr>
<tr>
<td></td>
<td>Q:F ratio</td>
<td>0.38</td>
</tr>
</tbody>
</table>
E.4.22 Brazilian Indirect Tensile Strength Horizontal Domainal Schistosity Sample

The goal of the horizontal domainal schistosity (DS) sample was to generate a fabric in which the mica grains are aligned, continuously connected across the sample, visible to the naked eye and anastamosing around feldspar and quartz microlithons.

Figure E.4.23: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

Table E.4.22: Input and output parameters from sample horizontal DS.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>Content</td>
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<tr>
<td>14.8</td>
<td></td>
</tr>
<tr>
<td>Mica</td>
<td>13</td>
</tr>
<tr>
<td>Quartz</td>
<td>24</td>
</tr>
<tr>
<td>Feldspar</td>
<td>63</td>
</tr>
<tr>
<td>Q:F ratio</td>
<td>0.38</td>
</tr>
</tbody>
</table>
**E.4.23 Brazilian Indirect Tensile Strength Type 1 Schistosity Sample**

The goal of the Type 1 schistosity (T1) sample was to generate a fabric in which the mica grains are aligned, continuously connected across the sample, visible to the naked eye but in which microlithons are not visible to the naked eye.

![Image](image_url)

Figure E.4.24: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
<th>Content</th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>Mica</td>
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<tr>
<td></td>
<td>Quartz</td>
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<td></td>
<td>Feldspar</td>
<td>62</td>
</tr>
<tr>
<td></td>
<td>Q:F ratio</td>
<td>0.27</td>
</tr>
</tbody>
</table>

Table E.4.23: Input and output parameters from sample horizontal T1.
E.4.24  Brazilian Indirect Tensile Strength Horizontal Type 1 Schistosity Sample

The goal of the horizontal Type 1 schistosity (T1) sample was to generate a fabric in which the mica grains are aligned, continuously connected across the sample, visible to the naked eye but in which microlithons are not visible to the naked eye.

Table E.4.24: Input and output parameters from sample horizontal T1.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
<th>Content</th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>Mica</td>
<td>23</td>
</tr>
<tr>
<td></td>
<td>Quartz</td>
<td>17</td>
</tr>
<tr>
<td></td>
<td>Feldspar</td>
<td>60</td>
</tr>
<tr>
<td></td>
<td>Q:F ratio</td>
<td>0.28</td>
</tr>
</tbody>
</table>
E.4.25  **Brazilian Indirect Tensile Strength Type 2 Schistosity Sample**

The goal of the Type 2 schistosity (T2) sample was to generate a fabric in which the mica grains are aligned, continuously connected across the sample, visible to the naked eye but in no microlithons are visible.

![Figure E.4.26: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.](image)

Table E.4.25: Input and output parameters from sample T2.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
<th>Content</th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
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</tr>
<tr>
<td></td>
<td>Quartz</td>
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<tr>
<td></td>
<td>Feldspar</td>
<td>49</td>
</tr>
<tr>
<td></td>
<td>Q:F ratio</td>
<td>0.47</td>
</tr>
</tbody>
</table>
**E.4.26 Brazilian Indirect Tensile Strength Horizontal Type 2 Schistosity Sample**

The goal of the horizontal Type 2 schistosity (T2) sample was to generate a fabric in which the mica grains are aligned, continuously connected across the sample, visible to the naked eye but in no microlithons are visible.

Figure E.4.27: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

Table E.4.26: Input and output parameters from sample horizontal T2.

<table>
<thead>
<tr>
<th><strong>Strength Indicators</strong></th>
<th><strong>Mineralogy</strong></th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>Content</td>
</tr>
<tr>
<td>13.7</td>
<td>Mica 30</td>
</tr>
<tr>
<td></td>
<td>Quartz 21</td>
</tr>
<tr>
<td></td>
<td>Feldspar 49</td>
</tr>
<tr>
<td></td>
<td>Q:F ratio 0.43</td>
</tr>
</tbody>
</table>
The goal of the domainal cleavage (DC) sample was to generate a fabric in which the mica grains are aligned, continuously connected across the sample, not visible to the naked eye and with a spacing between 0.5-5mm.

Table E.4.27: Input and output parameters from sample DC.

<table>
<thead>
<tr>
<th><strong>Strength Indicators</strong></th>
<th><strong>Mineralogy</strong></th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>14.3</td>
</tr>
<tr>
<td>Mica</td>
<td>19</td>
</tr>
<tr>
<td>Quartz</td>
<td>36</td>
</tr>
<tr>
<td>Feldspar</td>
<td>45</td>
</tr>
<tr>
<td>Q:F ratio</td>
<td>0.8</td>
</tr>
</tbody>
</table>
**E.4.28 Brazilian Indirect Tensile Strength Horizontal Dominal Cleavage (>5mm spacing) Sample**

The goal of the horizontal dominal cleavage (DC) sample was to generate a fabric in which the mica grains are aligned, continuously connected across the sample, not visible to the naked eye and with a spacing between 0.5-5mm.

![Image of failure maps and mineralogy](image.png)

Figure E.4.29: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

<table>
<thead>
<tr>
<th><strong>Strength Indicators</strong></th>
<th><strong>Mineralogy</strong></th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>Content</td>
</tr>
<tr>
<td>16.5</td>
<td></td>
</tr>
<tr>
<td>Mica</td>
<td>20</td>
</tr>
<tr>
<td>Quartz</td>
<td>34</td>
</tr>
<tr>
<td>Feldspar</td>
<td>46</td>
</tr>
<tr>
<td>Q:F ratio</td>
<td>0.74</td>
</tr>
</tbody>
</table>
### E.4.29 Brazilian Indirect Tensile Strength Continuous Cleavage (0.5-5mm spacing) Sample

The goal of the intermediate continuous cleavage (CC) sample was to generate a fabric in which the mica grains are aligned, continuously connected across the sample, not visible to the naked eye and with a spacing greater than 5mm.

![Image](image.jpg)

Figure E.4.30: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
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</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>10.9</td>
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<tr>
<td>Mica</td>
<td>27</td>
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<tr>
<td>Quartz</td>
<td>37</td>
</tr>
<tr>
<td>Feldspar</td>
<td>36</td>
</tr>
<tr>
<td>Q:F ratio</td>
<td>1.03</td>
</tr>
</tbody>
</table>

Table E.4.29: Input and output parameters from sample IC.
E.4.30  **Brazilian Indirect Tensile Strength Horizontal Continuous Cleavage (0.5-5mm spacing) Sample**

The goal of the horizontal intermediate continuous cleavage (CC) sample was to generate a fabric in which the mica grains are aligned, continuously connected across the sample, not visible to the naked eye and with a spacing greater than 5mm.

![Image of sample](image.png)

Figure E.4.31: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

<table>
<thead>
<tr>
<th><strong>Strength Indicators</strong></th>
<th><strong>Mineralogy</strong></th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>14.6</td>
</tr>
<tr>
<td><strong>Mineral</strong></td>
<td><strong>Content</strong></td>
</tr>
<tr>
<td>Mica</td>
<td>30</td>
</tr>
<tr>
<td>Quartz</td>
<td>35</td>
</tr>
<tr>
<td>Feldspar</td>
<td>35</td>
</tr>
<tr>
<td>Q:F ratio</td>
<td>1</td>
</tr>
</tbody>
</table>

Table E.4.30: Input and output parameters from sample horizontal IC.
E.4.31  Brazilian Indirect Tensile Strength Continuous Cleavage (<0.5mm spacing) Sample

The goal of the continuous cleavage (CC) sample was to generate a fabric in which the mica grains are aligned, continuously connected across the sample, not visible to the naked eye and with a spacing less than 0.5mm.

Figure E.4.32: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

Table E.4.31: Input and output parameters from sample CC.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
<th>Content</th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>Mica</td>
<td>28</td>
</tr>
<tr>
<td></td>
<td>Quartz</td>
<td>34</td>
</tr>
<tr>
<td></td>
<td>Feldspar</td>
<td>38</td>
</tr>
<tr>
<td></td>
<td>Q:F ratio</td>
<td>0.89</td>
</tr>
</tbody>
</table>
The goal of the horizontal continuous cleavage (CC) sample was to generate a fabric in which the mica grains are aligned, continuously connected across the sample, not visible to the naked eye and with a spacing less than 0.5mm.

Table E.4.32: Input and output parameters from sample CC horizontal.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
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</tr>
<tr>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
</tr>
</tbody>
</table>
The goal of the mineral preferred orientation widely spaced (MPOg) sample was to generate a fabric in which the mica grains are aligned, but not continuously connected across the sample, with a spacing greater than 5mm.

Figure E.4.34: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

Table E.4.33: Input and output parameters from sample MPOg.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Content</th>
<th>Mineralogy</th>
</tr>
</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
<td>16.4</td>
<td></td>
</tr>
<tr>
<td>Mica</td>
<td>10</td>
<td></td>
</tr>
<tr>
<td>Quartz</td>
<td>26</td>
<td></td>
</tr>
<tr>
<td>Feldspar</td>
<td>64</td>
<td></td>
</tr>
<tr>
<td>Q:F ratio</td>
<td>0.41</td>
<td></td>
</tr>
</tbody>
</table>
E.4.34 Brazilian Indirect Tensile Strength Horizontal Mineral Preferred Orientation Widely Spaced (>5mm mica spacing) Sample

The goal of the horizontal mineral preferred orientation widely spaced (MPOg) sample was to generate a fabric in which the mica grains are aligned, but not continuously connected across the sample, with a spacing greater than 5mm.

Figure E.4.35: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

Table E.4.34: Input and output parameters from sample horizontal MPOg.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
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</thead>
<tbody>
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<td>Brazilian Indirect Tensile (MPa)</td>
<td>Content</td>
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<td>18.9</td>
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<tr>
<td>Mica</td>
<td>10</td>
</tr>
<tr>
<td>Quartz</td>
<td>24</td>
</tr>
<tr>
<td>Feldspar</td>
<td>66</td>
</tr>
<tr>
<td>Q:F ratio</td>
<td>0.36</td>
</tr>
</tbody>
</table>
The goal of the mineral preferred orientation narrowly spaced (MPOI) sample was to generate a fabric in which the mica grains are aligned, but not continuously connected across the sample, with a spacing less than 5mm.

Table E.4.35: Input and output parameters from sample MPOI.

<table>
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<tr>
<th>Strength Indicators</th>
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</thead>
<tbody>
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<td>Content</td>
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<tr>
<td>13.4</td>
<td>Mica</td>
</tr>
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<tr>
<td></td>
<td>Quartz</td>
</tr>
<tr>
<td></td>
<td>22</td>
</tr>
<tr>
<td></td>
<td>Feldspar</td>
</tr>
<tr>
<td></td>
<td>65</td>
</tr>
<tr>
<td>Q:F ratio</td>
<td>0.34</td>
</tr>
</tbody>
</table>
The goal of the horizontal mineral preferred orientation narrowly spaced (MPOl) sample was to generate a fabric in which the mica grains are aligned, but not continuously connected across the sample, with a spacing less than 5mm.

Figure E.4.37: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.

Table E.4.36: Input and output parameters from sample horizontal MPOl.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
<th>Content</th>
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</thead>
<tbody>
<tr>
<td>Brazilian Indirect Tensile (MPa)</td>
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<td>16</td>
</tr>
<tr>
<td></td>
<td>Quartz</td>
<td>19</td>
</tr>
<tr>
<td></td>
<td>Feldspar</td>
<td>65</td>
</tr>
<tr>
<td></td>
<td>Q:F ratio</td>
<td>0.29</td>
</tr>
</tbody>
</table>
The goal of the mica variation 1 (M1) sample was to increase the mica content, while keeping the quartz to feldspar ratio the same. This sample has been classified as spalling sensitive due to the prevalence of tensile fracture propagation leading to failure.

Figure E.4.38: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.37: Input and output parameters from sample M1.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
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<tbody>
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<td>Initiation (MPa)</td>
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<td>Mica</td>
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<tr>
<td>Coalescence (MPa)</td>
<td>Quartz</td>
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<tr>
<td>Peak (MPa)</td>
<td>Feldspar</td>
</tr>
<tr>
<td>Failure Angle</td>
<td>Q:F ratio</td>
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<tr>
<td>Stiffness (GPa)</td>
<td></td>
</tr>
</tbody>
</table>

Figure E.4.39: Stress-strain and stress-AE graphs for sample M1.
The goal of the mica variation (M2) sample was to increase the mica content, while keeping the quartz to feldspar ratio the same. This sample has been classified as spalling insensitive due to the prevalence of shear fractures through feldspars and micas leading to shear failure surface development.
Table E.4.38: Input and output parameters from sample M2.

<table>
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</thead>
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<td>Initiation (MPa)</td>
<td>Content</td>
</tr>
<tr>
<td>Coalescence (MPa)</td>
<td>% failed</td>
</tr>
<tr>
<td>Peak (MPa)</td>
<td></td>
</tr>
<tr>
<td>Failure Angle</td>
<td></td>
</tr>
<tr>
<td>Stiffness (GPa)</td>
<td>Q:F ratio</td>
</tr>
<tr>
<td></td>
<td></td>
</tr>
<tr>
<td>60</td>
<td>17</td>
</tr>
<tr>
<td>105</td>
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<tr>
<td>138</td>
<td>15</td>
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<tr>
<td>76</td>
<td>68</td>
</tr>
<tr>
<td>53</td>
<td>0.22</td>
</tr>
</tbody>
</table>

Figure E.4.41: Stress-strain and stress-AE graphs for sample M2.
The goal of the mica variation (M3) sample was to increase the mica content, while keeping the quartz to feldspar ratio the same. This sample has been classified as spalling insensitive due to the prevalence of shear fractures through feldspars and micas leading to shear failure surface development.

Figure E.4.42: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.39: Input and output parameters from sample M3.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
<th>Content</th>
<th>% failed</th>
</tr>
</thead>
<tbody>
<tr>
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<td></td>
</tr>
<tr>
<td>Coalescence (MPa)</td>
<td>Mica</td>
<td>81</td>
<td>24</td>
</tr>
<tr>
<td>Peak (MPa)</td>
<td>Quartz</td>
<td>128</td>
<td>17</td>
</tr>
<tr>
<td>Failure Angle</td>
<td>Feldspar</td>
<td>50</td>
<td>50</td>
</tr>
<tr>
<td>Stiffness (GPa)</td>
<td>Q:F ratio</td>
<td>47</td>
<td>44</td>
</tr>
</tbody>
</table>

Figure E.4.43: Stress-strain and stress-AE graphs for sample M3.
The goal of the quartz variation (Q1) sample was to increase the quartz content, while keeping the mica to feldspar ratio the same. This sample has been classified as spalling insensitive due to the prevalence of shear fractures through feldspars and micas leading to shear failure surface development.

Figure E.4.44: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.40: Input and output parameters from sample Q1.

<table>
<thead>
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</tr>
</thead>
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<tr>
<td>Coalescence (MPa)</td>
<td>Mica</td>
<td>10</td>
<td>28</td>
</tr>
<tr>
<td>Peak (MPa)</td>
<td>Quartz</td>
<td>42</td>
<td>38</td>
</tr>
<tr>
<td>Failure Angle</td>
<td>Feldspar</td>
<td>48</td>
<td>33</td>
</tr>
<tr>
<td>Stiffness (GPa)</td>
<td>Q:F ratio</td>
<td>51</td>
<td>0.875</td>
</tr>
</tbody>
</table>

![Stress-strain and stress-AE graphs for sample Q1.](image)

Figure E.4.45: Stress-strain and stress-AE graphs for sample Q1.
E.4.41  UCS Quartz Variation 2 Sample

The goal of the quartz variation (Q2) sample was to increase the quartz content, while keeping the mica to feldspar ratio the same. This sample has been classified as spalling sensitive due to the prevalence of tensile fracture propagation leading to failure.

Figure E.4.46: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.41: Input and output parameters from sample Q2.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
<th>Content</th>
<th>% failed</th>
</tr>
</thead>
<tbody>
<tr>
<td>Initiation (MPa)</td>
<td></td>
<td>91</td>
<td></td>
</tr>
<tr>
<td>Coalescence (MPa)</td>
<td>Mica</td>
<td>150</td>
<td>6</td>
</tr>
<tr>
<td>Peak (MPa)</td>
<td>Quartz</td>
<td>181</td>
<td>18</td>
</tr>
<tr>
<td>Failure Angle</td>
<td>Feldspar</td>
<td>50</td>
<td>71</td>
</tr>
<tr>
<td>Stiffness (GPa)</td>
<td>Q:F ratio</td>
<td>51</td>
<td>35</td>
</tr>
</tbody>
</table>

Figure E.4.47: Stress-strain and stress-AE graphs for sample Q2.
**E.4.42  UCS Quartz Variation 3 Sample**

The goal of the quartz variation (Q3) sample was to increase the quartz content, while keeping the mica to feldspar ratio the same. This sample has been classified as spalling sensitive due to the prevalence of tensile fracture propagation leading to failure.

Figure E.4.48: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.42: Input and output parameters from sample Q3.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
<th>Content</th>
<th>% failed</th>
</tr>
</thead>
<tbody>
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<td></td>
</tr>
<tr>
<td>Coalescence (MPa)</td>
<td></td>
<td>213</td>
<td>Mica</td>
</tr>
<tr>
<td>Peak (MPa)</td>
<td></td>
<td>3</td>
<td>238</td>
</tr>
<tr>
<td>Failure Angle</td>
<td></td>
<td>85</td>
<td>Quartz</td>
</tr>
<tr>
<td>Stiffness (GPa)</td>
<td></td>
<td>71</td>
<td>84</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>12</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>67</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>51</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>Q:F ratio 7.1</td>
</tr>
</tbody>
</table>

Figure E.4.49: Stress-strain and stress-AE graphs for sample Q3.
The goal of the quartz variation (Q35) sample was to increase the quartz content, while keeping the mica content very low. This sample has been classified as spalling sensitive due to the prevalence of tensile fracture propagation leading to failure.

Figure E.4.50: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.43: Input and output parameters from sample Q35.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
</tr>
<tr>
<td>Initiation (MPa)</td>
<td>89</td>
</tr>
<tr>
<td>Coalescence (MPa)</td>
<td>169</td>
</tr>
<tr>
<td>Peak (MPa)</td>
<td>196</td>
</tr>
<tr>
<td>Failure Angle</td>
<td>35</td>
</tr>
<tr>
<td>Stiffness (GPa)</td>
<td>50</td>
</tr>
</tbody>
</table>

Figure E.4.51: Stress-strain and stress-AE graphs for sample Q35.
The goal of the quartz variation (Q36) sample was to increase the quartz content, while keeping the mica content very low. This sample has been classified as spalling sensitive due to the prevalence of tensile fracture propagation leading to failure.

Figure E.4.52: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.44: Input and output parameters from sample Q36.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
<th>Content</th>
<th>% failed</th>
</tr>
</thead>
<tbody>
<tr>
<td>Initiation (MPa)</td>
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<td>135</td>
<td></td>
</tr>
<tr>
<td>Coalescence (MPa)</td>
<td>Mica</td>
<td>208</td>
<td></td>
</tr>
<tr>
<td>Peak (MPa)</td>
<td>Quartz</td>
<td>249</td>
<td>93</td>
</tr>
<tr>
<td>Failure Angle</td>
<td>Feldspar</td>
<td>69</td>
<td>99</td>
</tr>
<tr>
<td>Stiffness (GPa)</td>
<td>Q:F ratio</td>
<td>49</td>
<td>84</td>
</tr>
</tbody>
</table>

Figure E.4.53: Stress-strain and stress-AE graphs for sample Q36.
The goal of the quartz variation 4 (Q4) sample was to increase the quartz to feldspar ratio, while keeping the mica content the same. This sample has been classified as spalling insensitive due to the prevalence of shear fractures through feldspars and micas leading to shear failure surface development.

Figure E.4.54: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.45: Input and output parameters from sample Q4.

<table>
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<tr>
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<th>Mineralogy</th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Initiation (MPa)</td>
<td>56</td>
<td>Mica</td>
<td>12</td>
<td>30</td>
</tr>
<tr>
<td>Coalescence (MPa)</td>
<td>106</td>
<td>Quartz</td>
<td>15</td>
<td>38</td>
</tr>
<tr>
<td>Peak (MPa)</td>
<td>147</td>
<td>Feldspar</td>
<td>73</td>
<td>27</td>
</tr>
<tr>
<td>Failure Angle</td>
<td>66</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Stiffness (GPa)</td>
<td>55</td>
<td>Q:F ratio</td>
<td>0.21</td>
<td></td>
</tr>
</tbody>
</table>

Figure E.4.55: Stress-strain and stress-AE graphs for sample Q4.
The goal of the quartz variation (Q5) sample was to increase the quartz to feldspar ratio, while keeping the mica content the same. This sample has been classified as spalling sensitive due to the prevalence of tensile fracture propagation leading to failure.

Figure E.4.56: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.46: Input and output parameters from sample Q5.

<table>
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<th>Strength Indicators</th>
<th>Mineralogy</th>
</tr>
</thead>
<tbody>
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<td>Initiation (MPa)</td>
<td>Content</td>
</tr>
<tr>
<td>56</td>
<td>Mica</td>
</tr>
<tr>
<td>Coalescence (MPa)</td>
<td></td>
</tr>
<tr>
<td>96</td>
<td>Mica</td>
</tr>
<tr>
<td>Peak (MPa)</td>
<td></td>
</tr>
<tr>
<td>154</td>
<td>Quartz</td>
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<tr>
<td>Failure Angle</td>
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<td>47</td>
<td>Feldspar</td>
</tr>
<tr>
<td>Stiffness (GPa)</td>
<td></td>
</tr>
<tr>
<td>51</td>
<td>Q:F ratio</td>
</tr>
</tbody>
</table>

![Stress-strain and stress-AE graphs for sample Q5.](image)

Figure E.4.57: Stress-strain and stress-AE graphs for sample Q5.
**E.4.47 UCS Quartz Variation 6 sample**

The goal of the quartz variation (Q6) sample was to increase the quartz to feldspar ratio, while keeping the mica content the same. This sample has been classified as spalling sensitive due to the prevalence of tensile fracture propagation leading to failure.

Figure E.4.58: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.47: Input and output parameters from sample Q6.

<table>
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</thead>
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<td>Content % failed</td>
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<td>6</td>
</tr>
<tr>
<td>Coalescence (MPa)</td>
<td>29</td>
</tr>
<tr>
<td>104</td>
<td>Mica</td>
</tr>
<tr>
<td>Peak (MPa)</td>
<td>19</td>
</tr>
<tr>
<td>164</td>
<td>Quartz</td>
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<tr>
<td>Failure Angle</td>
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<tr>
<td>72</td>
<td>Feldspar</td>
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<td>Stiffness (GPa)</td>
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</tr>
<tr>
<td>47</td>
<td>Q:F ratio</td>
</tr>
<tr>
<td>3.45</td>
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Figure E.4.59: Stress-strain and stress-AE graphs for sample Q6.
The goal of the quartz variation (Q65) sample was to increase the quartz to feldspar ratio, while keeping the mica content the same. This sample has been classified as spalling sensitive due to the prevalence of tensile fracture propagation leading to failure.

Figure E.4.60: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.48: Input and output parameters from sample Q65.

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<td>Failure Angle</td>
<td>Feldspar</td>
<td>75</td>
<td>55</td>
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<td>Stiffness (GPa)</td>
<td>Q:F ratio</td>
<td>45</td>
<td>10.13</td>
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![Stress-strain and stress-AE graphs for sample Q65.](image)
The goal of the quartz variation (Q7) sample was to increase the quartz to feldspar ratio, while keeping the mica content the same. This sample has been classified as spalling sensitive due to the prevalence of tensile fracture propagation leading to failure.

Figure E.4.62: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.49: Input and output parameters from sample Q7.

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<td>Peak (MPa)</td>
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<tr>
<td>Failure Angle</td>
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<tr>
<td>Stiffness (GPa)</td>
<td>44</td>
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<td>Quartz</td>
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<td>Stiffness (GPa)</td>
<td>Q:F ratio</td>
<td>21</td>
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</table>

Figure E.4.63: Stress-strain and stress-AE graphs for sample Q7.
E.4.50 **UCS Grain Size 1 No Grain Boundaries (0.5mm grain size) Sample**

The goal of the grain size 1 no grain boundaries (G1NB) sample was to specify the grain size without grain boundaries while keeping the mineralogy the same. This sample has been classified as spalling insensitive due to the prevalence of shear fracture propagation leading to the development of shear failure planes.

Figure E.4.64: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.50: Input and output parameters from sample G1NB.

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<tr>
<td>Peak (MPa)</td>
<td>210 Quartz</td>
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<tr>
<td>Failure Angle</td>
<td>67 Feldspar</td>
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<tr>
<td>Stiffness (GPa)</td>
<td>71 Q:F ratio</td>
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<tr>
<td>Content</td>
<td>9 53</td>
</tr>
<tr>
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<td>21 66</td>
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<td>70 62</td>
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</table>

Figure E.4.65: Stress-strain and stress-AE graphs for sample G1NB.
E.4.51  **UCS Grain Size 2 No Grain Boundaries (4mm grain size) Sample**

The goal of the grain size 2 no grain boundaries (G2NB) sample was to specify the grain size without grain boundaries while keeping the mineralogy the same. This sample has been classified as spalling insensitive due to the prevalence of shear fracture propagation in feldspars and micas leading to the development of shear failure planes, as well as the plastic deformation during yielding as shown in Figure E.3.66.

Figure E.4.66: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.51: Input and output parameters from sample G2NB.

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<tr>
<td>Peak (MPa)</td>
<td>Quartz</td>
</tr>
<tr>
<td>Failure Angle</td>
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</tr>
<tr>
<td>Stiffness (GPa)</td>
<td>74</td>
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<tr>
<td>Content</td>
<td>8</td>
</tr>
<tr>
<td>Q:F ratio</td>
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</table>

Figure E.4.67: Stress-strain and stress-AE graphs for sample G2NB.
E.4.52  

**UCS Grain Size 3 No Grain Boundaries (8mm grain size) Sample**

The goal of the grain size 3 no grain boundaries (G3NB) sample was to specify the grain size without grain boundaries while keeping the mineralogy the same. This sample has been classified as spalling sensitive due to the prevalence of tensile fracture propagation during sample failure.

Figure E.4.68: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.52: Input and output parameters from sample G3NB.

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<td>Peak (MPa)</td>
<td>Quartz</td>
<td>178</td>
<td>9</td>
</tr>
<tr>
<td>Failure Angle</td>
<td>Feldspar</td>
<td>68</td>
<td>72</td>
</tr>
<tr>
<td>Stiffness (GPa)</td>
<td>Q:F ratio</td>
<td>74</td>
<td>0.28</td>
</tr>
</tbody>
</table>

Figure E.4.69: Stress-strain and stress-AE graphs for sample G3NB.
The goal of the grain size 2 (G2) sample was to generate a sample with an average grain size of 4mm with grain boundaries one element thick. This sample has been classified as spalling insensitive due to the prevalence of shear fracture propagation during development of a shear failure plane and the plastic deformation during yielding, as shown in Figure E.4.70.

Figure E.4.70: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.53: Input and output parameters from sample G2.

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<td>Quartz</td>
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<td>Failure Angle</td>
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<tr>
<td>Stiffness (GPa)</td>
<td>Q:F ratio</td>
<td>56</td>
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Figure E.4.71: Stress-strain and stress-AE graphs for sample G2.
The goal of the grain size 3 (G3) sample was to generate a sample with an average grain size of 8mm with grain boundaries one element thick. This sample has been classified as spalling sensitive due to the prevalence of tensile fracture propagation during sample failure.

Figure E.4.72: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.54: Input and output parameters from sample G3.

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<td>Q:F ratio</td>
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Figure E.4.73: Stress-strain and stress-AE graphs for sample G3.
E.4.55  **UCS Grain Size 4 (12mm with grain boundaries)**

**Sample**

The goal of the grain size 4 (G4) sample was generate a sample with an average grain size of 12mm with grain boundaries one element thick. This sample has been classified as spalling sensitive due to the prevalence of tensile fracture propagation during sample failure.

Figure E.4.74: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.55: Input and output parameters from sample G4.

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</tr>
<tr>
<td>124</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Peak (MPa)</td>
<td>Quartz</td>
<td>133</td>
<td></td>
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<tr>
<td>133</td>
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<td></td>
<td></td>
</tr>
<tr>
<td>Failure Angle</td>
<td>Feldspar</td>
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<tr>
<td>55-90</td>
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<td>65</td>
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Figure E.4.75: Stress-strain and stress-AE graphs for sample G4.
E.4.56  **UCS Grain Size 5 (16mm with grain boundaries)**

Sample

The goal of the grain size 5 (G5) sample was generate a sample with an average grain size of 16mm with grain boundaries one element thick. This sample has been classified as spalling sensitive due to the prevalence of tensile fracture propagation during the generation of a shear failure plane.

Figure E.4.76: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.56: Input and output parameters from sample G5.

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<td>Coalescence (MPa)</td>
<td>Quartz</td>
</tr>
<tr>
<td>Peak (MPa)</td>
<td>Feldspar</td>
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<tr>
<td>Failure Angle</td>
<td>Q:F ratio</td>
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<td>Stiffness (GPa)</td>
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Figure E.4.77: Stress-strain and stress-AE graphs for sample G5.
E.4.57  **UCS Domainal Schistosity Sample**

The goal of the domainal schistosity (DS) sample was to generate a fabric in which the mica grains are aligned, continuously connected across the sample, visible to the naked eye and anastamosing around feldspar and quartz microlithons. This sample contains aspects of tensile and shear fracture, but the prevalence of tensile fracture during sample failure suggests it is sensitive to spalling.

Figure E.4.78: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.3.1.
Table E.4.57: Input and output parameters from sample DS.

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<td>Peak (MPa)</td>
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<td>Failure Angle</td>
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<td>Stiffness (GPa)</td>
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</table>

Figure E.4.79: Stress-strain and stress-AE graphs for sample DS.
**E.4.58 UCS Type 1 Schistosity Sample**

The goal of the Type 1 schistosity (T1) sample was to generate a fabric in which the mica grains are aligned, continuously connected across the sample, visible to the naked eye but in which microlithons are not visible to the naked eye. This sample contains aspects of tensile and shear fracture, but the prevalence of shear fracture in the development of a clear shear failure surface suggests it is insensitive to spalling.

Figure E.4.80: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.58: Input and output parameters from sample T1.

<table>
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<td>Coalescence (MPa)</td>
<td>Content</td>
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<td>Quartz</td>
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<td>Failure Angle</td>
<td>Feldspar</td>
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<td>Stiffness (GPa)</td>
<td>Q:F ratio</td>
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</table>

Figure E.4.81: Stress-strain and stress-AE graphs for sample T1.
E.4.59 **UCS Type 2 Schistosity Sample**

The goal of the Type 2 schistosity (T2) sample was to generate a fabric in which the mica grains are aligned, continuously connected across the sample, visible to the naked eye but in no microlithons are visible. This sample contains aspects of tensile and shear fracture, but the prevalence of tensile fracture in the sample failure suggests it is sensitive to spalling.

Figure E.4.82: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.59: Input and output parameters from sample T2.

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<td>Mica</td>
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<td>28</td>
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<tr>
<td>Coalescence (MPa)</td>
<td>Mica</td>
<td>28</td>
<td>42</td>
</tr>
<tr>
<td>Peak (MPa)</td>
<td>Quartz</td>
<td>124</td>
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<tr>
<td>Failure Angle</td>
<td>Feldspar</td>
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<td>50</td>
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<td>Stiffness (GPa)</td>
<td>Q:F ratio</td>
<td>44</td>
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</table>

Figure E.4.83: Stress-strain and stress-AE graphs for sample T2.
The goal of the continuous cleavage (CC) sample was to generate a fabric in which the mica grains are aligned, continuously connected across the sample, not visible to the naked eye and with a spacing less than 0.5mm. This sample contains aspects of tensile and shear fracture, but the prevalence of shear fracture in the feldspar and mica grains in the generation of a shear failure surface suggests it is insensitive to spalling.

Figure E.4.88: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.62: Input and output parameters from sample CC.

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</tr>
<tr>
<td>Coalescence (MPa)</td>
<td>Quartz</td>
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<td>Peak (MPa)</td>
<td>Feldspar</td>
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</tr>
<tr>
<td>Stiffness (GPa)</td>
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</table>

Figure E.4.89: Stress-strain and stress-AE graphs for sample CC.
The goal of the intermediate continuous cleavage (CC) sample was to generate a fabric in which the mica grains are aligned, continuously connected across the sample, not visible to the naked eye and with a spacing greater than 5mm. This sample contains aspects of tensile and shear fracture, but the prevalence of shear fracture in the feldspar and mica grains in the generation of a shear failure surface suggests it is insensitive to spalling.

Figure E.4.86: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.61: Input and output parameters from sample CC.

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<td>Mica</td>
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<tr>
<td>Peak (MPa)</td>
<td>Quartz</td>
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<td>Feldspar</td>
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<tr>
<td>Stiffness (GPa)</td>
<td>Q:F ratio</td>
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</table>

Figure E.4.87: Stress-strain and stress-AE graphs for sample CC.
E.4.60  **UCS Domainal Cleavage (>5mm spacing) Sample**

The goal of the domainal cleavage (DC) sample was to generate a fabric in which the mica grains are aligned, continuously connected across the sample, not visible to the naked eye and with a spacing between 0.5-5mm. This sample contains aspects of tensile and shear fracture, but the prevalence of shear fracture in the feldspar grains in the generation of a shear failure surface suggests it is insensitive to spalling.

![Image of mineralogy](image)

Figure E.4.84: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.60: Input and output parameters from sample DC.

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<td>28</td>
</tr>
<tr>
<td>Coalescence (MPa)</td>
<td>Quartz</td>
<td>33</td>
<td>53</td>
</tr>
<tr>
<td>Peak (MPa)</td>
<td>Feldspar</td>
<td>47</td>
<td>49</td>
</tr>
<tr>
<td>Failure Angle</td>
<td>Q:F ratio</td>
<td>0.70</td>
<td></td>
</tr>
<tr>
<td>Stiffness (GPa)</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Figure E.4.85: Stress-strain and stress-AE graphs for sample DC.
**E.4.63 UCS Mineral Preferred Orientation Widely Spaced (>5mm mica spacing) Sample**

The goal of the mineral preferred orientation widely spaced (MPOg) sample was to generate a fabric in which the mica grains are aligned, but not continuously connected across the sample, with a spacing greater than 5mm. This sample contains mainly aspects of tensile and some shear failure, but tensile fracture leading to failure makes it sensitive to spalling.

Figure E.4.90: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.63: Input and output parameters from sample MPOg.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
<th>Content</th>
<th>% failed</th>
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</thead>
<tbody>
<tr>
<td>Initiation (MPa)</td>
<td></td>
<td>49</td>
<td></td>
</tr>
<tr>
<td>Coalescence (MPa)</td>
<td>Mica</td>
<td>105</td>
<td>11</td>
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<tr>
<td>Peak (MPa)</td>
<td>Quartz</td>
<td>129</td>
<td>25</td>
</tr>
<tr>
<td>Failure Angle</td>
<td>Feldspar</td>
<td>67</td>
<td>64</td>
</tr>
<tr>
<td>Stiffness (GPa)</td>
<td>Q:F ratio</td>
<td>56</td>
<td>29</td>
</tr>
</tbody>
</table>

Figure E.4.91: Stress-strain and stress-AE graphs for sample MPOg.
The goal of the mineral preferred orientation narrowly spaced (MPO) sample was to generate a fabric in which the mica grains are aligned, but not continuously connected across the sample, with a spacing less than 5mm. This sample contains aspects of tensile and shear failure, but the prevalence of tensile fracture in the generation of a shear failure surface suggests it is sensitive to spalling.

Figure E.4.92: From top-right: failure map at 75% of peak strength, failure map and strain map at 95% of peak strength, failure map and strain map post peak, and image of mineralogy. See legend in Figure E.4.1.
Table E.4.64: Input and output parameters from sample MPOI.

<table>
<thead>
<tr>
<th>Strength Indicators</th>
<th>Mineralogy</th>
<th>Content</th>
<th>% failed</th>
</tr>
</thead>
<tbody>
<tr>
<td>Initiation (MPa)</td>
<td>Mica</td>
<td>13</td>
<td>33</td>
</tr>
<tr>
<td>Coalescence (MPa)</td>
<td>Quartz</td>
<td>24</td>
<td>38</td>
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<tr>
<td>Peak (MPa)</td>
<td>Feldspar</td>
<td>63</td>
<td>26</td>
</tr>
<tr>
<td>Failure Angle</td>
<td>Q:F ratio</td>
<td>0.38</td>
<td></td>
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<tr>
<td>Stiffness (GPa)</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Figure E.4.93: Stress-strain and stress-AE graphs for sample MPOI.
Appendix F

This appendix contains documents related directly to the material presented in Chapter 6.
Appendix F.1 : Calibration and Development of Two-Cutter Models

F.1.1 Introduction

This appendix contains the background, development and sensitivity calibration of the two cutter numerical models. These models were created to simulate the rock cutting process under two adjacent cutters. The models were then used to investigate the impact of different geomechanical characteristics and in-situ stress conditions on the cutting process. This section contains a description of the model geometry, loading methodology, and sensitivity analysis to element size, model size and configuration of the cutters and boundaries.

F.1.2 Large-Scale Two-Cutter Model

The large-scale two-cutter model (Figure F.1.1) is a cross sectional model of two 43cm diameter cutters, 9cm apart centre on centre, that are forced into a 2.5x1.2m block of rock. The block of rock is made up of two grids: a larger exterior grid that reduces in size towards a smaller interior grid. This is in order to save computational and memory effort, while satisfying the exterior boundaries and interior grid requirements. The smaller interior grid contains 5mm wide elements. The exterior grid nodes match exactly with the interior grid nodes.

The grid nodes at the base of the cutters also match exactly with the nodes in the interior rock block grid. This was done in order to minimise instability due to incompatible grids and to ensure that the forces are transmitted directly from one grid to another. The cutters are attached to the rock grid to ensure that they are in contact with the surface of the rock block grid during loading. If they were not attached, the cutters would simply move in space without any effect on the rock block.

The vertical exterior boundaries of the large scale grid of the rock block and the cutters are fixed in the x direction so that they are only allowed to deform and/or move vertically. The lower horizontal boundary of the large-scale grid of the rock block is fixed in the y direction so that it is only allowed to deform horizontally. The corners of the large-scale grid of the rock block are fixed in x and y directions to ensure that the entire model does not move in space. The interior grid of the rock block is attached to the exterior grid to ensure contact between the two grids and to prevent movement between them.
Sensitivity analysis of the element size was not undertaken as the element size used in this model is the smallest that was computationally realistic. The small grid element size is one order of magnitude bigger than the UCS, Brazilian and fracture toughness models and sensitivity analysis of larger elements is unnecessary, while analysis of smaller elements would be too computationally intensive.

Figure F.1.1: Schematic of large-scale two-cutter model showing dimensions and boundary conditions (grid not to scale).
**F.1.3 Large-Scale Model Velocity Function for Cutter Movement**

A velocity function is applied to the cutters (Figure F.1.1) to provide the vertical movement necessary to simulate the chipping process. In three dimensions the total force acting on a cutter (and, consequently, the rock) includes a normal vertical force and a horizontal rolling force (Figure F.1.2). The model was created in two-dimensions, as an axial cross-section, precluding the need to simulate the rolling force.

The cutting process is achieved by rolling a cutter along the surface of a tunnel wall. This means that the reaction force in a particular block of rock due to the cutting process is cyclical, arising from the arrival, presence and departure of the cutter. The model simulates a particular block of rock and the impact of the arrival, presence and departure of two adjacent cutters.

The arrival, presence and departure are simulated using a number of steps:

1. The model initially only includes the rock block
2. The left cutter is generated
3. A velocity is applied following a sinusoidal function to simulate the arrival, presence and departure of the cutter
4. The left cutter is removed
5. The model is cycled without any cutters present to simulate the delay between the departure of one cutter and the arrival of the adjacent cutter
6. Steps 2-4 are repeated for the right cutter

The sinusoidal function used to simulate the arrival, presence and departure of the cutter uses this formulation:

\[
V = \left( \sin \left( \frac{n}{n_o} \right) \right) s \quad \text{F.1.1}
\]

where \( V \) = velocity, \( n \) = step number in model, \( n_o \) = initial step number and \( s \) = scaling factor.

The initial step number is varied to provide the spread of steps over which the velocity will be applied while the scaling factor is varied to provide the magnitude of the velocity.
Figure F.1.2: Schematic of the cutter forces during cutter excavation (after Rostami and Ozdemir (1993))

Figure F.1.3: Graph of the sigmoidal function used to generate the velocity of the cutter
In order to determine the magnitude of velocity applied to the cutter, a series of model runs were undertaken to iteratively determine the velocity required to result in the required force at the cutter/rock interface. The force magnitude required is approximately $6.7 \times 10^6$ N. This value is based on the maximum force allowed for 43cm diameter cutters.

In TBM application the maximum normal force on a cutter is approximately 263kN. This force is distributed over an elliptical contact area resulting in an average stress at the contact. In two dimensions the same stress must be distributed over a contact strip 1m long normal to the section and with the actual width in the section. In order to relate the maximum normal force in three dimensions to two dimensions the stress at the cutter tip in three dimensions was determined based on the surface area of cutter contact in a 2mm deep kerf (see Figure F.1.4 for schematic). The surface area was estimated as the length of chord x width of cutter. The stress at the tip of a cutter in three dimensions was estimated as the maximum force divided by the contact area (Table F.1.1). The resulting stress was then multiplied by the contact area in the 2D model (0.03m$^2$) to obtain the force at the cutter tip of $6.7 \times 10^6$N. The model was run with elastic properties to calibrate for the stress magnitude (Figure F.1.5). The result of the application of this velocity function to the top of the cutters in the mohr-coulomb model is shown in Figures F.1.6-F.1.8.

<table>
<thead>
<tr>
<th>Dimensions</th>
<th>Contact Area ($m^2$)</th>
<th>Force at tip (N)</th>
<th>Stress at tip (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Three</td>
<td>$1.172 \times 10^{-3}$</td>
<td>2.63e5</td>
<td>220</td>
</tr>
<tr>
<td>Two</td>
<td>$3 \times 10^{-2}$</td>
<td>6.7e6</td>
<td>220</td>
</tr>
</tbody>
</table>

Table F.1.1: Summary of contact area, force and stress at the tip of the three and two-dimensional cutters
Figure F.1.4: Schematic of the dimensions used to determine the surface area of contact of a three dimensional cutter. Cutter width is 20mm.

Figure F.1.5: Output of elemental stress at the cutter/rock interface resulting from the applied velocity functions (left first, then right) in the elastic model.
Figure F.1.6: Output of force at the cutter/rock interface resulting from the applied velocity functions (left first, then right).

Figure F.1.7: Output of elemental stress at the cutter/rock interface resulting from the applied velocity functions (left first, then right).
The resulting depth of cutter penetration (Figure F.1.7) shows that the model deforms within the area of the cutters since there is some vertical deformation recorded for the rock area below the right cutter when the left cutter is loaded, and vice versa. The order of magnitude of cutter penetration is on the $10^{-1}$ mm scale, which is one order of magnitude lower than penetration observed during TBM excavation (typically between 3-10 mm per cutterhead revolution). The difference arises because the displacement in the model is due to deformation of the elements, related to their stiffness, whereas the displacement during excavation is due to the generation and removal of fines (at low penetration depth) or chips (at higher penetration depth).

The left and right cutter loading and unloading cycle is repeated four times to simulate four rotations of a TBM cutterhead. This number of cycles was selected because it is felt that more than one cycle is needed to pre-condition the rock: that is generate some fractures in the rock since in actuality the TBM is never thrusting against pristine rock, rather it is always thrusting against pre-existing kerfs. It is also felt that three cycles is sufficient to generate the preconditioning (first cycle) and observe the impact of subsequent cutter applications.
F.1.4 Large-Scale Model Homogeneity, Heterogeneity and In-situ Stress Condition

Several constitutive models were used in the two-cutter model, each with progressively more complex heterogeneity to investigate the differences in element failure arising from increasing heterogeneity in the modelled rock. In addition, an in-situ stress condition was applied to each of the progressively more heterogeneous models to again investigate the impact of stress on the failure patterns (Figure F.1.9).

The original constitutive model used in the two-cutter model is a homogeneous strain-softening material with the same parameters as the homogeneous model in Appendix E.2. Figure F.1.10 shows the locations of element failures in the homogeneous models with and without stress. The failure is limited to the immediate contact area between the cutters and the rock and does not differ considerably whether there is or is not in-situ stress.

Figure F.1.9: FLAC model outputs demonstrating applied in-situ stresses (left) at model boundary and interior stresses in model (right, magnitude=30Mpa)

Figure F.1.10: FLAC model outputs of homogeneous model showing failure locations in homogeneous model without (left) and with (right) applied in-situ stress.
The model with higher heterogeneity used the same heterogeneous constitutive model as in Appendix 4.2, with 3.25% random variability in the input parameters on an element by element basis (Figure F.1.11). Figure F.1.12 shows the locations of element failures in the heterogeneous models with and without stress. The failure is limited to the immediate contact area between the cutters and the rock and does not differ considerably whether there is or is not in-situ stress, and is very similar to the homogeneous model.

Figure F.1.11: FLAC model outputs showing variability in the heterogeneous model

Figure F.1.12: FLAC model outputs of heterogeneous model showing failure locations in homogeneous model without (left) and with (right) applied in-situ stress.
The texture generation algorithm developed in Appendix D.2 was used in the two-cutter model (Figure F.1.13) to investigate the impact of explicit heterogeneity and fabric alignment on the failure pattern. Figure F.1.14 shows the failure pattern for the schistosity type 1 material. The failure is no longer restricted to the boundary and some of the failed elements follow the direction of the fabric. Figure F.1.15 shows the failure pattern for the same material with in-situ stress. The failure pattern is much more distributed and less concentrated at the cutter/rock interface. In this case the in-situ stress appears to inhibit fracture propagation, and microfractures initiated in the rock are blunted and remain disconnected, but act to precondition the rock. The mesh used in these models is 10 times bigger than the mesh used in the UCS and Brazilian models, which also used the texture algorithm, regardless, it demonstrates that adding heterogeneity in the form of a fabric provides more realistic failure results and will allow the investigation of impacts of different fabric types on the disc cutting process, similar to the investigation on UCS and Brazilian undertaken in Chapter 5.

Figure F.1.13: FLAC model outputs showing fabric, in this case schistosity type 1, aligned at 60° to the tunnel face boundary. Red = quartz, yellow = feldspar, teal = mica.
Figure F.1.14: FLAC model outputs of schistosity type 1 fabric in Figure F.1.13 showing element failure pattern (left) and strain (right).

Figure F.1.15: FLAC model outputs of schistosity type 1 fabric in Figure F.1.12 with applied in-situ stress showing element failure pattern (left) and strain (right).

For illustration purposes, the fabric from Figure F.1.13 was oriented at 0° to the boundary to investigate the impact of parallel fabric alignment on the failure pattern (Figure F.1.16). Figure F.1.17 shows the failure pattern is similar to the failure pattern in Figure F.1.14, with a few more failed elements, but still restricted to the cutter area. Figures F.1.18 and F.1.19 show the failure pattern for the same material with in-situ stress, which is similar to the failure pattern in Figure F.1.15, again with more failed elements acting to precondition the rock parallel to the fabric and, consequently, to the excavation boundary. In this case the in-situ stress appears to inhibit fracture
propagation into the rock, but promotes fracture parallel to the boundary, generating a chip between the cutters.

Figure F.1.16: FLAC model outputs showing fabric, in this case schistosity type 1, aligned at 0° to the tunnel face boundary. Red = quartz, yellow = feldspar, teal = mica.

Figure F.1.17: FLAC model outputs of schistosity type 1 fabric in Figure F.1.16 showing element failure pattern (left) and strain (right).
F.1.5 Large-Scale Model Kerf Simulation

An improvement to the simulation of progressive cutter excavation was made to simulate the advance of the cutter into the excavated rock with each subsequent cutter pass. In the kerf simulation the failed elements with direct access to the excavation boundary are ‘removed’ (Figure F.1.20) to simulate the removal of excavated material (at this point material that has been ground). In the place of the failed material steel is created below the cutters to simulate the
subsequent cutter pass in the deeper kerf (Figure F.1.21). Figures F.1.22 to F.1.25 show the model outputs from the two-cutter model with type 1 schistosity and the kerf simulation algorithm with and without applied stress. The results shown in Figure F.1.23 are similar to those shown in Figure F.1.14 in terms of location, although more elements failed using the kerf simulation algorithm. The failure pattern shown in Figure F.1.23 consists of several adjacent failed elements that define a macrofracture resembling the type of fracture that leads to chip formation between cutters. The results shown in Figure F.1.25 are similar to those shown in Figure F.1.15 in terms of location of failure and the distributed failed elements throughout the model, although more elements failed near the cutters using the kerf simulation algorithm. The failure pattern shown in Figure F.1.25 has failed elements that also resemble the type of fracture that leads to chip formation between cutters, similar to the results in Figure F.1.23.

The kerf simulation algorithm provides results that more closely simulate the progression of the cutter into the rock as it excavates. The algorithm simulates the material removed due to crushing and grinding directly under and adjacent to the cutter and leads to fracture patterns resembling chip formation between the cutters. The addition of this algorithm to the two-cutter model improves the results and will be used in the parametric analysis.

---

Figure F.1.20: Schematic of large-scale two-cutter model showing removal of failed elements adjacent to excavation boundary to simulate kerf creation (grid not to scale). Steel shown in black.
Figure F.1.21: Schematic of large-scale two-cutter model showing addition of steel into kerf created in Figure F.1.19 to simulate cutters advancing into the newly excavated kerf (grid not to scale). Steel shown in black.

Figure F.1.22: FLAC model output of schistosity type 1 fabric in Figure F.1.12 with kerf generation showing element failure pattern (left) and strain (right).
Figure F.1.23: FLAC model outputs of schistosity type 1 fabric in Figure F.1.12 with kerf generation showing location of newly generated cutters inside the kerfs.

Figure F.1.24: FLAC model output of schistosity type 1 fabric in Figure F.1.12 with applied in-situ stress showing element failure pattern (left) and strain (right).
The grid in the fine-grid section of the large-scale cutter model is made up of 5mm elements. This is 10 times larger than the grid used in the Brazilian and UCS modelling undertaken in Chapter 5. In order to create a model with the same element size, thus making the work from Chapter 5 comparable to the two-cutter modelling, the element size was reduced by one order of magnitude. This fine grid could not be generated with the same geometry as the large-scale cutter model due to the resulting number of elements and the exorbitant computation time required to run such large models.

The new small-scale grid is shown in Figure F.1.26 and has 0.5mm elements in the rock block. Each cutter is represented by half a cutter and the rock block is 90mm wide by 120mm deep. The boundary conditions include rollers at the cutter sides and rock block bottom and sides, and pins at each corner to anchor the system. The velocity function applied to the cutters was determined by the same methodology as the large-scale model (Figure F.1.27). The resulting force and elemental stress in the mohr-coulomb model are shown in Figures F.1.28 and F.1.29 while the deformation of the grid due to cutter application is shown in Figure F.1.30. The stress below the cutters in the small-scale model are similar in magnitude to the stresses below the cutters in the large-scale model (Figure F.1.6), while the depth of deformation is less in the small-scale model than in the large-scale model (Figure F.1.7). This difference likely arises from the small deformation possible in the smaller elements of the small-scale model.
The investigations with the homogeneous and homogeneous-variable rock material were not undertaken for the small-scale model as it was shown for the large-scale model that rock simulated with the texture algorithm results in a more realistic simulation of the cutting process. Instead, sensitivity to the rock block dimensions and boundary conditions were investigated using rock simulated by the texture algorithm.

Figure F.1.26: Left: Schematic of small-scale two-cutter model showing dimensions and boundary conditions (grid not to scale); right: FLAC output of the small-scale two-cutter model.
Figure F.1.27: Output of elemental stress at the cutter/rock interface resulting from the applied velocity functions (left first, then right) in the elastic model.

Figure F.1.28: Output of force at the cutter/rock interface resulting from the applied velocity functions (left first, then right).
Figure F.1.29: Output of elemental stress at the cutter/rock interface resulting from the applied velocity functions (left first, then right).

Figure F.1.30: Output of displacement (in metres) at the cutter/rock interface resulting from the applied velocity functions (left first, then right).
F.1.7 Small-Scale Model Boundary Conditions

The small-scale two-cutter model was run with and without applied stress with the schistosity type 1 texture (Figure F.1.31). The failure patterns for the model without applied stress after 1 cutter pass are shown in Figure F.1.32 and after three passes in Figure F.1.33. The failure patterns for the model with applied stress after 1 cutter pass are shown in Figure F.1.34 and after three passes in Figure F.1.35. The models are not vastly different after the third cutter pass compared to the first cutter pass.

The failure pattern at the cutter/rock interface and between the cutters differs from the results from the large-scale model in Figures F.1.14 and F.1.15. These differences may in part arise from the difference in grid size: the large grid size in the large-scale model may allow more adjacent failed elements that approach the simulation of chip formation and failed elements reach a greater depth into the rock block. The smaller grid size in the small-scale model allow a better interpretation of the failure pattern and allows a more realistic simulation of rock texture due to the finer resolution of failed elements.

The rollers at the base of the rock block were replaced by pins (fixed in both the x and y directions) to simulate the lateral confinement that would arise from the surrounding rock and prevent the propagation of a median crack between cutters (Figure F.1.34). A median crack would not actually form in confined rock and arises solely due to the small size of the rock block and the rollers allowing unhindered lateral deformation and never occurred with the large-scale model. No median crack formed in the models depicted in Figures F.1.32 and F.1.33, and the results shown in Figure F.1.34 do not considerably differ, suggesting that the nature of the horizontal boundary (pins or rollers) is not critical to the failure pattern. For this reason, rollers will be used for subsequent models.
Figure F.1.31: FLAC small-scale model outputs showing fabric, in this case schistosity type 1, aligned at 60° to the tunnel face boundary. Red = quartz, yellow = feldspar, teal = mica.

Figure F.1.32: FLAC model outputs of schistosity type 1 fabric in Figure F.1.31 showing element failure pattern after one cutter pass (left) and strain (right).
F.1.8 Small-Scale Model Rock-Block Size

The sensitivity to the rock block size was investigated by lengthening the rock block to 150mm and 180mm (Figures F.1.35 and F.1.36, respectively). The results vary only slightly from the results shown in Figure F.1.33, suggesting that a rock block 90x120mm in dimensions is large enough for this application.
Figure F.1.35: FLAC model outputs of schistosity type 1 fabric in Figure F.1.31 with rock block dimensions 90x150mm showing element failure pattern (left) and strain (right) after three cutter passes.

Figure F.1.36: FLAC model outputs of schistosity type 1 fabric in Figure F.1.31 with rock block dimensions 90x180mm showing element failure pattern (left) and strain (right) after three cutter passes.

F.1.9 Small-Scale Model In-Situ Stress Condition

Figure F.1.37 shows the failure pattern for the same material with in-situ stress. The failure pattern is much more distributed and less concentrated at the cutter/rock interface. In this case the in-situ stress appears to inhibit fracture propagation, and microfractures initiated in the rock are blunted and remain disconnected, but act to precondition the rock in a similar way as was shown with the large-scale two-cutter model.
**F.1.10 Small-Scale Model Kerf Simulation**

The kerf simulation methodology was also applied to the small-scale two-cutter model to simulate the progressive penetration of the cutters (Figures F.1.38 to F.1.41). The results using the kerf simulation are similar to their counterparts without the kerf simulation in Figures F.1.33 and F.1.34, although more failed elements are concentrated near the cutter-rock interface. This simulates the progressive grinding of the material below the cutter as it advances through the rock.
Figure F.1.38: FLAC model output of schistosity type 1 fabric in Figure F.1.29 with kerf generation showing element failure pattern and accumulated strains (left) and strain (right).

Figure F.1.39: FLAC model outputs of schistosity type 1 fabric in Figure F.1.29 with kerf generation showing location of newly generated cutters inside the kerfs.
F.1.11 Small-Scale Model with Joint Interface

The difference in failure patterns between the large-scale and small-scale models, in particular the lack of chip formation in the small-scale model, was investigated by changing some of the boundary conditions in the small-scale model. This involved creating a joint interface between the cutter and the rock, rather than using a contiguous grid for both the rock and cutter. This joint has no shear or tensile strength, but does exhibit frictional properties to simulate the
independent steel cutter and rock block. The joint formulation in FLAC (Martin, 1997) requires normal and shear stiffness based on the stiffness of the adjacent material and the element width according to:

\[
\text{max} \left( \frac{K + \frac{4}{3}G}{z} \right)
\]

where K and G and bulk and shear modulus, respectively and z is the element width. Max refers to using the maximum value if surrounding elements have different stiffness moduli or element sizes. The joint stiffness moduli were calculated as 5e13 using this formula, based on the input parameters from Chapter 5 and the element width of 0.5mm.

The kerf algorithm was considerably modified for the joint boundary condition since the joint interface was required to progressively move with the advancing cutter. In addition, the kerf algorithm was simplified to simulate the advance of the cutter regardless of yielded elements to reflect the abrasive removal of rock material that is not directly modelled in this model. A nominal kerf depth was selected for cutter advance to represent abrasion based on identification of the minimum penetration rate observed in active TBM advance. This value was selected as 1mm, as seen in Figure F.1.42, and the cutter was advanced by 1mm for each cutter pass (Figure F.1.43). This is shallower than the kerf assumed in Table F.1.1, and the cutter tip stresses were recalculated using a 1mm kerf depth to be 316MPa (Figure F.1.44 and Table F.1.2). The use of a joint at the cutter-rock interface created realistic localized failure in a chip-like shape (Figure F.1.45), making it possible to distinguish chipping performance.
Figure F.1.42: Start-up tests from Amsteg TBM in Southern Aar granite.

Figure F.1.43: Schematic of cutter advance in 1mm increments to simulate 1mm deep kerf. The rock grid was cut during grid generation to allow the subsequent progression of the cutters (left to right). The joints subsequently used along cutter boundary (with joint properties) were temporarily attached, then given joint properties at the appropriate time. Other joints not used along cutter boundary (but necessary in grid generation) were attached throughout the simulation.
Figure F.1.44: Output of elemental stress at the cutter/rock interface resulting from the applied velocity functions (left first, then right) in the elastic model.

Table F.1.2: Summary of contact area, force and stress at the tip of the three and two-dimensional cutters with a 1mm deep kerf

<table>
<thead>
<tr>
<th>Dimensions</th>
<th>Contact Area (m²)</th>
<th>Force at tip (N)</th>
<th>Stress at tip (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Three</td>
<td>8.3x10⁻⁴</td>
<td>2.63e5</td>
<td>316</td>
</tr>
<tr>
<td>Two</td>
<td>3x10⁻²</td>
<td>9.4e6</td>
<td>316</td>
</tr>
</tbody>
</table>

Figure F.1.45: FLAC model outputs of schistosity type 1 fabric in Figure F.1.16 with applied in-situ stress showing element failure pattern overlaid on mineralogy (left) and strain (right).
**F.1.12 Two-Cutter Model for Parametric Analysis**

Based on the sensitivity analysis of the small-scale model it was found that the geometry shown in Figure F.1.43 results in the most realistic cutting behaviour and is most comparable to the UCS and Brazilian models without compromising computation time. The kerf simulation algorithm best simulates the generation and removal of crushed material adjacent to the cutters and is used in the parametric analysis.
Appendix F.2 Parametric Analysis of Chipping Performance and Face Stability Results

F.2.1 Introduction

This appendix contains the results of the parametric analysis of the two-cutter excavation system, as well as a classification for goodness of chip. The same parameters investigated with the UCS and Brazilian tensile strength models were used in model results without stress and are presented first. Within this category models with different fabric types were modeled with in-situ stress only to examine the impact on preconditioning. Models with varying stress conditions and fabric at different angles are presented second, and models representing the stress and geological conditions encountered in the Southern Aar granite and Altkristallin are presented third.

F.2.2 Goodness of Chip Designation

Due to computation time limitations complete chips were never obtained during parametric analysis. The plastic strains were used to examine the localization of failure, and each sample was categorised for goodness of chip according to (Figure F.2.1):

1. Clear localization of strain leading to chip formation
2. Localization of strain with bifurcation
3. Multiple zones of localized strain
4. Localized strain controlled by fabric, can be associated with lack of localized strain in adjacent cutter
5. Lack of clear localized strain

Categories 1-3 will have different lengths of localized strain zones depending on progression of the chip-defining fracture, which is interpreted as improved chipping. Each of the categories 1-3 were also given a percentage chip length based on the percentage of the potential chip that was actually manifested in the test. In addition to categorizing the chip type, the chip length and depth were measured from the strain plots (actively straining fractures are assumed to best represent chip) and used to calculate a chip area. The chip area allowed quantification of the goodness of chip for parametric analysis.
F.2.3 Two-Cutter Modelling without Stress

The samples modelled in this section are the same as those presented in Appendix E.4. The quartz variations 35 and 5 were not included because their characteristics and UCS and Brazilian tensile strength results were repetitive. In addition, the grain size models without grain boundaries were not modelled as they are not representative of real minerals. The materials are not described here, but the goodness of chip is discussed for each sample type. The gneissic fabric is composed of a band of dark minerals (mica and feldspar) and light minerals (quartz and feldspar) as shown in Figure F.2.2. Two different models were created for gneissic fabric: close to cutters and far from cutters, referring to the distance of the dark mineral band to the cutters. This simulates not only the impact of the dark band on the cutters, but also narrow and wide band spacing (where close to cutters simulates narrow spacing and far from cutters simulates wide spacing).
F.2.3.1 Mica Variation 1

The mica variation 1 sample is classified as a type 2 chip with 1cm chip length and 1.2cm chip depth on the left side and 0.8cm chip length and 1cm chip depth on the right side.
F.2.3.2 Mica Variation 2

The mica variation 2 sample is classified as a type 2 chip with 1cm chip length and 0.8cm chip depth on the left side and 1.5cm chip length and 1.8cm chip depth on the right side.

Figure F.2.4: FLAC output of deformed grid (left) and plastic shear strain (right) for mica variation 2.

F.2.3.3 Mica Variation 3

The mica variation 3 sample is classified as a type 2 chip with 1.5cm chip length and 1cm chip depth on the left side and 1.2cm chip length and 2cm chip depth on the right side.

Figure F.2.5: FLAC output of deformed grid (left) and plastic shear strain (right) for mica variation 3.
F.2.3.4 Quartz Variation 1

The quartz variation 1 sample is classified as a type 2 chip with 1.5cm chip length and 2cm chip depth on the left side and 1cm chip length and 2cm chip depth on the right side.

Figure F.2.6: FLAC output of deformed grid (left) and plastic shear strain (right) for quartz variation 1.

F.2.3.5 Quartz Variation 2

The quartz variation 2 sample is classified as a type 2 chip with 2cm chip length and 0.6cm chip depth on the left side and 0.4cm chip length and 1.5cm chip depth on the right side.

Figure F.2.7: FLAC output of deformed grid (left) and plastic shear strain (right) for quartz variation 2.
F.2.3.6 Quartz Variation 3

The quartz variation 3 sample is classified as a type 2 chip with 3cm chip length and 0.6cm chip depth on the left side and 1.5cm chip length and 0.3cm chip depth on the right side.

Figure F.2.8: FLAC output of deformed grid (left) and plastic shear strain (right) for quartz variation 3.

F.2.3.7 Quartz Variation 36

The quartz variation 36 sample is classified as a type 2 chip with 1.5cm chip length and 0.3cm chip depth on the left side and 1.3cm chip length and 0.4cm chip depth on the right side.

Figure F.2.9: FLAC output of deformed grid (left) and plastic shear strain (right) for quartz variation 36.
F.2.3.8 Quartz Variation 4

The quartz variation 4 sample is classified as a type 2 chip with 1cm chip length and 1cm chip depth on the left side and 1cm chip length and 0.3cm chip depth on the right side.

Figure F.2.10: FLAC output of deformed grid (left) and plastic shear strain (right) for quartz variation 4.

F.2.3.9 Quartz Variation 6

The quartz variation 6 sample is classified as a type 2 chip with 1.5cm chip length and 0.5cm chip depth on the left side and 0.5cm chip length and 0.5cm chip depth on the right side.

Figure F.2.11: FLAC output of deformed grid (left) and plastic shear strain (right) for quartz variation 6.
F.2.3.10 Quartz Variation 65

The quartz variation 65 sample is classified as a type 2 chip with 1cm chip length and 0.3cm chip depth on the left side and 1.3cm chip length and 0.2cm chip depth on the right side.

Figure F.2.12: FLAC output of deformed grid (left) and plastic shear strain (right) for quartz variation 6.

F.2.3.11 Quartz Variation 7

The quartz variation 7 sample is classified as a type 2 chip with 0.5cm chip length and 0.8cm chip depth on the left side and 1cm chip length and 1cm chip depth on the right side.

Figure F.2.13: FLAC output of deformed grid (left) and plastic shear strain (right) for quartz variation 7.
F.2.3.12 Grain Size 2

The grain size 2 sample is classified as a type 2 chip with 1cm chip length and 2cm chip depth on the left side and 1.3cm chip length and 0.8cm chip depth on the right side.

Figure F.2.14: FLAC output of deformed grid (left) and plastic shear strain (right) for grain size 2.

F.2.3.13 Grain Size 3

The grain size 3 sample is classified as a type 2 chip with 3cm chip length and 0.6cm chip depth on the left side and 1.6cm chip length and 1 cm chip depth on the right side.

Figure F.2.15: FLAC output of deformed grid (left) and plastic shear strain (right) for grain size 3.
**F.2.3.14 Grain Size 4**

The grain size 4 sample is classified as a type 2 chip with 4cm chip length and 1.5cm chip depth on the left side and 2.3cm chip length and 0.6cm chip depth on the right side.

Figure F.2.16: FLAC output of deformed grid (left) and plastic shear strain (right) for grain size 4.

**F.2.3.15 Grain Size 5**

The grain size 5 sample is classified as a type 2 chip with 6.5cm chip length and 1cm chip depth on the left side and 2cm chip length and 0.3cm chip depth on the right side.

Figure F.2.17: FLAC output of deformed grid (left) and plastic shear strain (right) for grain size 5.
F.2.3.16 Domainal Schistosity

The domainal schistosity sample is classified as a type 2 chip with 2.2cm chip length and 1.5cm chip depth on the left side and 1cm chip length and 1cm chip depth on the right side.

Figure F.2.18: FLAC output of deformed grid (left) and plastic shear strain (right) for domainal schistosity.

F.2.3.17 Type 1 Schistosity

The type 1 schistosity sample is classified as a type 2 chip with 1cm chip length and 1cm chip depth on the left side and 1cm chip length and 1.5cm chip depth on the right side.

Figure F.2.19: FLAC output of deformed grid (left) and plastic shear strain (right) for type 1 schistosity.
F.2.3.18 Type 2 Schistosity

The type 2 schistosity sample is classified as a type 3 chip with 1.3cm chip length and 0.5cm chip depth on the left side and 1.2cm chip length and 1.2cm chip depth on the right side.

Figure F.2.20: FLAC output of deformed grid (left) and plastic shear strain (right) for type 1 schistosity.

F.2.3.19 Domainal Cleavage

The domainal cleavage sample is classified as a type 2 chip with 1cm chip length and 1.2cm chip depth on the left side and 0.8cm chip length and 0.6cm chip depth on the right side.

Figure F.2.21: FLAC output of deformed grid (left) and plastic shear strain (right) for domainal cleavage.
**F.2.3.20 Continuous Cleavage (<0.5mm spacing)**

The continuous cleavage sample is classified as a type 2 chip with 1cm chip length and 0.8cm chip depth on the left side and 1.2cm chip length and 1.2cm chip depth on the right side.

![FLAC output of deformed grid (left) and plastic shear strain (right) for continuous cleavage](image1)

**F.2.3.21 Mineral Preferred Orientation Widely Spaced**

The widely spaced mineral preferred orientation sample is classified as a type 2 chip with 1.5cm chip length and 0.8cm chip depth on the left side and 1.1cm chip length and 0.5cm chip depth on the right side.

![FLAC output of deformed grid (left) and plastic shear strain (right) for MPO.](image2)
F.2.3.22 Gneissic Fabric Close to Cutters

The gneissic fabric close to cutters sample is classified as a type 2 chip with 2cm chip length and 1cm chip depth on the left side and 2.2cm chip length and 1.2cm chip depth on the right side.

Figure F.2.24: FLAC output of deformed grid (left) and plastic shear strain (right) for gneiss.

F.2.3.23 Gneissic Fabric Far from Cutters

The gneissic fabric far from cutters sample is classified as a type 2 chip with 1.2cm chip length and 0.2cm chip depth on the left side and 0.8cm chip length and 0.2cm chip depth on the right side.

Figure F.2.25: FLAC output of deformed grid (left) and plastic shear strain (right) for gneiss.
F.2.4 Parametric Modelling of Samples with Stress

The samples modelled in this section are some isotropic rocks and the fabric types at 60° to the tunnel face and intensities under high biaxial stress (30MPa horizontal and 50MPa vertical). Only the interaction between stress and fabric was modelled to determine magnitude of preconditioning and the cutters were not applied to these models. Images of failed elements and plastic strain are shown to demonstrate preconditioning.

F.2.4.1 Mica Variation 1

The mica variation 1 sample has mild preconditioning.

Figure F.2.26: FLAC output of element failure (left) and plastic shear strain (right) for mica variation 1.

F.2.4.2 Mica Variation 3

The mica variation 3 sample has moderate preconditioning.

Figure F.2.27: FLAC output of element failure (left) and plastic shear strain (right) for mica variation 3.
F.2.4.3 Quartz Variation 3

The quartz variation 3 sample has mild preconditioning.

Figure F.2.28: FLAC output of element failure (left) and plastic shear strain (right) for quartz variation 3.

F.2.4.4 Quartz Variation 6

The quartz variation 6 sample has moderate preconditioning.

Figure F.2.29: FLAC output of element failure (left) and plastic shear strain (right) for quartz variation 6.
F.2.4.5 Grain Size 2

The grain size 2 sample has moderate preconditioning.

Figure F.2.30: FLAC output of element failure (left) and plastic shear strain (right) for grain size 2.

F.2.4.6 Grain Size 4

The grain size 4 sample has moderate preconditioning.

Figure F.2.31: FLAC output of element failure (left) and plastic shear strain (right) for grain size 4.
F.2.4.7  Domainal Schistosity Stress Only

The domainal schistosity sample has moderate preconditioning.

Figure F.2.32: FLAC output of element failure (left) and plastic shear strain (right) for domainal schistosity.

F.2.4.8  Schistosity Type 1 Stress Only

The schistosity type 1 sample has mild preconditioning.

Figure F.2.33: FLAC output of element failure (left) and plastic shear strain (right) for schistosity type 1.
F.2.4.9 Schistosity Type 2 Stress Only

The schistosity type 1 sample has high preconditioning.

Figure F.2.34: FLAC output of element failure (left) and plastic shear strain (right) for schistosity type 2.

F.2.4.10 Domainal Cleavage Stress Only

The domainal cleavage sample has high preconditioning.

Figure F.2.35: FLAC output of element failure (left) and plastic shear strain (right) for domainal cleavage.
F.2.4.11 Continuous Cleavage Stress Only

The continuous cleavage sample has high preconditioning.

Figure F.2.36: FLAC output of element failure (left) and plastic shear strain (right) for continuous cleavage.

F.2.4.12 Mineral Preferred Orientation Stress Only

The mineral preferred orientation sample has moderate preconditioning.

Figure F.2.37: FLAC output of element failure (left) and plastic shear strain (right) for MPO.
F.2.4.13 Gneissic Fabric Stress Only

The gneiss sample has high preconditioning in the dark gneissic band (mica and feldspar) but low preconditioning in the light gneissic band (quartz and feldspar).

Figure F.2.38: FLAC output of element failure (left) and plastic shear strain (right) for MPO.

F.2.5 Two-Cutter Modelling with Stress

Several variations on fabric orientation and stress conditions are presented in this section. Subsections F.2.4.1 to F.2.4.4 have the same biaxial stress condition (Biaxial 50MPa horizontal and 30MPa vertical), while the fabric orientation with respect to the tunnel face was varied. Subsections F.2.4.5 to F.2.4.16 model eight different stress conditions with fabric oriented at 60° and 0° to the tunnel face. The eight stress conditions are described in Table F.2.1.

Table F.2.1: Summary of in-situ stress conditions

<table>
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<th>Stress state</th>
<th>In-Situ Stress (Input)</th>
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<td>Stress parallel</td>
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<td></td>
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<td>to face (MPa)</td>
<td>stress (MPa)</td>
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<tr>
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<tr>
<td>No Damage Isotropic low</td>
<td>0</td>
<td>20</td>
<td>20</td>
</tr>
<tr>
<td>No Damage Isotropic medium</td>
<td>0</td>
<td>35</td>
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</tr>
<tr>
<td>No Damage Isotropic high</td>
<td>0</td>
<td>55</td>
<td>55</td>
</tr>
<tr>
<td>No Damage Biaxial very low</td>
<td>0</td>
<td>10</td>
<td>5</td>
</tr>
<tr>
<td>No Damage Biaxial low</td>
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<tr>
<td>No Damage Biaxial medium</td>
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<td>35</td>
<td>17</td>
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<tr>
<td>No Damage Biaxial high</td>
<td>0</td>
<td>55</td>
<td>27</td>
</tr>
</tbody>
</table>

**F.2.5.1 Fabric 90° to the Tunnel Face**

The fabric 90° to the tunnel face sample is classified as a type 4 chip with 1cm chip length and 0.5cm chip depth on the left side and 0.5cm chip length and 0.4cm chip depth on the right side.

![Figure F.2.39: FLAC output of deformed grid (left) and plastic shear strain (right) for fabric 90° to tunnel face.](image)
F.2.5.2 Fabric 60° to the Tunnel Face

The fabric 60° to the tunnel face sample is classified as a type 1 chip with 0.8cm chip length and 1.2cm chip depth on the left side and 1cm chip length and 1.5cm chip depth on the right side.

Figure F.2.40: FLAC output of deformed grid (left) and plastic shear strain (right) for fabric 60° to tunnel face.

F.2.5.3 Fabric 30° to the Tunnel Face

The fabric 30° to the tunnel face sample is classified as a type 1 chip with 2.5cm chip length and 0.8cm chip depth on the left side and 1.5cm chip length and 1cm chip depth on the right side.

Figure F.2.41: FLAC output of deformed grid (left) and plastic shear strain (right) for fabric 30° to tunnel face.
F.2.5.4 Fabric $0^\circ$ to the Tunnel Face

The fabric $0^\circ$ to the tunnel face sample is classified as a type 1 chip with 0.8cm chip length and 0.4cm chip depth on the left side and 1cm chip length and 0.5cm chip depth on the right side.

![Figure F.2.42: FLAC output of deformed grid (left) and plastic shear strain (right) for fabric $0^\circ$ to tunnel face.](image)

F.2.5.5 Very Low Isotropic Stress with Fabric $60^\circ$ to the Tunnel Face

The very low isotropic stress with fabric $60^\circ$ to the tunnel face sample is classified as a type 1 chip with 1cm chip length and 0.3cm chip depth on the left side and 0.9cm chip length and 0.3cm chip depth on the right side.

![Figure F.2.43: FLAC output of deformed grid (left) for low isotropic stress with fabric $60^\circ$ to tunnel face.](image)
F.2.5.6 Very Low Isotropic Stress with Fabric 0° to the Tunnel Face

The low isotropic stress with fabric 0° to the tunnel face sample is classified as a type 2 chip with 1.2cm chip length and 0.3cm chip depth on the left side and 2.6cm chip length and 0.7cm chip depth on the right side.

![Figure F.2.44: FLAC output of deformed grid (left) and plastic shear strain (right) for low isotropic stress with fabric 0° to tunnel face.]

F.2.5.7 Very Low Isotropic Stress without Fabric

The low isotropic stress without fabric sample is classified as a type 2 chip with 1cm chip length and 1cm chip depth on the left side and 0.6cm chip length and 1cm chip depth on the right side.

![Figure F.2.45: FLAC output of deformed grid (left) and plastic shear strain (right) for low isotropic stress without fabric.]

F.2.5.8  Low Isotropic Stress with Fabric 60° to the Tunnel Face

The low isotropic stress with fabric 60° to the tunnel face sample is classified as a type 2 chip with 1.2cm chip length and 1.2cm chip depth on the left side and 1.2cm chip length and 1.5cm chip depth on the right side.

Figure F.2.46: FLAC output of deformed grid (left) and plastic shear strain (right) for low isotropic stress with fabric 60° to tunnel face.

F.2.5.9  Low Isotropic Stress with Fabric 0° to the Tunnel Face

The low isotropic stress with fabric 0° to the tunnel face sample is classified as a type 1 chip with 2cm chip length and 0.8cm chip depth on the left side and 1.5cm chip length and 0.4cm chip depth on the right side.

Figure F.2.47: FLAC output of deformed grid (left) and plastic shear strain (right) for low isotropic stress with fabric 0° to tunnel face.
**F.2.5.10 Medium Isotropic Stress with Fabric 60° to the Tunnel Face**

The medium isotropic stress with fabric 60° to the tunnel face sample is classified as a type 4 chip with 1cm chip length and 0.8cm chip depth on the left side and 0.8cm chip length and 0.5cm chip depth on the right side.

![FLAC output of deformed grid (left) and plastic shear strain (right) for medium isotropic stress with fabric 60° to tunnel face.](image)

**F.2.5.11 Medium Isotropic Stress with Fabric 0° to the Tunnel Face**

The medium isotropic stress with fabric 0° to the tunnel face sample is classified as a type 1 chip with 2.1cm chip length and 0.8cm chip depth on the left side and 1.7cm chip length and 0.4cm chip depth on the right side.

![FLAC output of deformed grid (left) and plastic shear strain (right) for medium isotropic stress with fabric 0° to tunnel face.](image)
F.2.5.12 High Isotropic Stress with Fabric 60° to the Tunnel Face

The high isotropic stress with fabric 60° to the tunnel face sample is classified as a type 4 chip with 1cm chip length and 0.3cm chip depth on the left side and 1.2cm chip length and 1.2cm chip depth on the right side.

Figure F.2.50: FLAC output of deformed grid (left) and plastic shear strain (right) for high isotropic stress with fabric 60° to tunnel face.

F.2.5.13 High Isotropic Stress with Fabric 0° to the Tunnel Face

The high isotropic stress with fabric 0° to the tunnel face sample is classified as a type 1 chip with 0.5cm chip length and 0.4cm chip depth on the left side and 3.2cm chip length and 0.4cm chip depth on the right side.

Figure F.2.51: FLAC output of deformed grid (left) and plastic shear strain (right) for high isotropic stress with fabric 0° to tunnel face.
F.2.5.14 Low Biaxial Stress with Fabric 60° to the Tunnel Face

The low biaxial stress with fabric 60° to the tunnel face sample is classified as a type 1 chip with 1.5cm chip length and 0.5cm chip depth on the left side and 0.5cm chip length and 1cm chip depth on the right side.

![Figure F.2.52: FLAC output of deformed grid (left) and plastic shear strain (right) for low biaxial stress with fabric 60° to tunnel face.](image)

F.2.5.15 Low Biaxial Stress with Fabric 0° to the Tunnel Face

The low biaxial stress with fabric 0° to the tunnel face sample is classified as a type 2 chip with 2cm chip length and 0.5cm chip depth on the left side and 2.1cm chip length and 1cm chip depth on the right side.

![Figure F.2.53: FLAC output of deformed grid (left) and plastic shear strain (right) for low biaxial stress with fabric 0° to tunnel face.](image)
F.2.5.16 Medium Biaxial Stress with Fabric 60° to the Tunnel Face

The medium biaxial stress with fabric 60° to the tunnel face sample is classified as a type 4 chip with 1cm chip length and 0.8cm chip depth on the left side and 0.8cm chip length and 0.5cm chip depth on the right side.

Figure F.2.54: FLAC output of deformed grid (left) and plastic shear strain (right) for medium biaxial stress with fabric 60° to tunnel face.

F.2.5.17 Medium Biaxial Stress with Fabric 0° to the Tunnel Face

The medium biaxial stress with fabric 0° to the tunnel face sample is classified as a type 1 chip with 2.2cm chip length and 0.8cm chip depth on the left side and 1.6cm chip length and 0.5cm chip depth on the right side.

Figure F.2.55: FLAC output of deformed grid (left) and plastic shear strain (right) for medium biaxial stress with fabric 0° to tunnel face.
F.2.5.18 High Biaxial Stress with Fabric 60° to the Tunnel Face

The high biaxial stress with fabric 60° to the tunnel face sample is classified as a type 1 chip with 1.5cm chip length and 0.8cm chip depth on the left side and 2cm chip length and 1.3cm chip depth on the right side.

Figure F.2.56: FLAC output of deformed grid (left) and plastic shear strain (right) for high biaxial stress with fabric 60° to tunnel face.

F.2.5.19 High Biaxial Stress with Fabric 0° to the Tunnel Face

The high biaxial stress with fabric 0° to the tunnel face sample is classified as a type 1 chip with 1cm chip length and 0.9cm chip depth on the left side and 2.5cm chip length and 1cm chip depth on the right side.

Figure F.2.57: FLAC output of deformed grid (left) and plastic shear strain (right) for high biaxial stress with fabric 0° to tunnel face.
**F.2.5.20 Biaxial Low Stress Ratio Perpendicular to Face with Fabric 60° to the Tunnel Face**

The biaxial with low stress ratio perpendicular to face with fabric 60° to the tunnel face sample is classified as a type 2 chip with 1.5cm chip length and 0.8cm chip depth on the left side and 1.5cm chip length and 1.5cm chip depth on the right side.

![Figure F.2.58: FLAC output of deformed grid (left) and plastic shear strain (right) for biaxial low stress ratio with fabric 60° to tunnel face](image)

**F.2.5.21 Biaxial Low Stress Ratio Perpendicular to Face with Fabric 0° to the Tunnel Face**

The biaxial with low stress ratio perpendicular to face with fabric 0° to the tunnel face sample is classified as a type 1 chip with 1.5cm chip length and 0.8cm chip depth on the left side and 1cm chip length and 0.4cm chip depth on the right side.

![Figure F.2.59: FLAC output of deformed grid (left) and plastic shear strain (right) for biaxial low stress ratio with fabric 0° to tunnel face.](image)
**F.2.5.22 Biaxial High Stress Ratio Perpendicular to Face with Fabric 60° to the Tunnel Face**

The biaxial with high stress ratio perpendicular to face with fabric 60° to the tunnel face sample is classified as a type 3 chip with 1.5cm chip length and 0.5cm chip depth on the left side and 1.2cm chip length and 1.2cm chip depth on the right side.

![Figure F.2.60: FLAC output of deformed grid (left) and plastic shear strain (right) for biaxial high stress ratio with fabric 60° to tunnel face.](image)

**F.2.5.23 Biaxial High Stress Ratio Perpendicular to Face with Fabric 0° to the Tunnel Face**

The biaxial with high stress ratio perpendicular to face with fabric 0° to the tunnel face sample is classified as a type 3 chip with 1.2cm chip length and 1.2cm chip depth on the left side and 3cm chip length and 1cm chip depth on the right side.

![Figure F.2.61: FLAC output of deformed grid (left) and plastic shear strain (right) for biaxial high stress ratio with fabric 0° to tunnel face.](image)
F.2.5.24 Biaxial Low Stress Ratio Parallel to Face with Fabric 60° to the Tunnel Face

The biaxial with low stress ratio parallel to face with fabric 60° to the tunnel face sample is classified as a type 1 chip with 1cm chip length and 0.5cm chip depth on the left side and 0.6cm chip length and 0.3cm chip depth on the right side.

Figure F.2.62: FLAC output of deformed grid (left) and plastic shear strain (right) for biaxial low stress ratio parallel to face with fabric 60° to tunnel face.

F.2.5.25 Biaxial Low Stress Ratio Parallel to Face with Fabric 0° to the Tunnel Face

The biaxial with low stress ratio parallel to face with fabric 0° to the tunnel face sample is classified as a type 4 chip with 1.2cm chip length and 0.8cm chip depth on the left side and 0.4cm chip length and 0.2cm chip depth on the right side.

Figure F.2.63: FLAC output of deformed grid (left) and plastic shear strain (right) for biaxial low stress ratio parallel to face with fabric 0° to tunnel face.
F.2.5.26 Biaxial Medium Stress Ratio Parallel to Face with Fabric 60° to the Tunnel Face

The biaxial with medium stress ratio parallel to face with fabric 60° to the tunnel face sample is classified as a type 1 chip with 1.2cm chip length and 0.3cm chip depth on the left side and 1.2cm chip length and 0.3cm chip depth on the right side.

![Figure F.2.64: FLAC output of deformed grid (left) and plastic shear strain (right) for biaxial medium stress ratio parallel to face with fabric 60° to tunnel face.]

F.2.5.27 Biaxial Medium Stress Ratio Parallel to Face with Fabric 0° to the Tunnel Face

The biaxial with medium stress ratio parallel to face with fabric 0° to the tunnel face sample is classified as a type 4 chip with 1.2cm chip length and 0.8cm chip depth on the left side and 0.4cm chip length and 0.2cm chip depth on the right side.

![Figure F.2.65: FLAC output of deformed grid (left) and plastic shear strain (right) for biaxial medium stress ratio parallel to face with fabric 0° to tunnel face.]

767
F.2.5.28 Isotropic Very Low Stress No Damage with Fabric 45° to the Tunnel Face

The isotropic very low stress no damage with fabric 45° to the tunnel face sample is classified as a type 4 chip with 1cm chip length and 0.6cm chip depth on the left side and 1.1cm chip length and 1.8cm chip depth on the right side.

Figure F.2.66: FLAC output of deformed grid (left) and plastic shear strain (right) for isotropic very low stress no damage with fabric 45° to tunnel face.

F.2.5.29 Isotropic Very Low Stress No Damage with Fabric 0° to the Tunnel Face

The isotropic very low stress no damage with fabric 0° to the tunnel face sample is classified as a type 4 chip with 2.6cm chip length and 0.8cm chip depth on the left side and 0.6cm chip length and 0.2cm chip depth on the right side.

Figure F.2.67: FLAC output of deformed grid (left) and plastic shear strain (right) for isotropic very low stress no damage with fabric 0° to tunnel face.
**F.2.5.30 Isotropic Very Low Stress No Damage without Fabric**

The isotropic very low stress no damage without fabric sample is classified as a type 3 chip with 2cm chip length and 0.7cm chip depth on the left side and 1.6cm chip length and 0.7cm chip depth on the right side.

![FLAC output of deformed grid (left) and plastic shear strain (right) for isotropic very low stress no damage without fabric.](image)

**F.2.5.31 Isotropic Low Stress No Damage with Fabric 45° to the Tunnel Face**

The isotropic low stress no damage with fabric 45° to the tunnel face sample is classified as a type 2 chip with 0.6cm chip length and 0.6cm chip depth on the left side and 0.5cm chip length and 0.3cm chip depth on the right side.

![FLAC output of deformed grid (left) and plastic shear strain (right) for isotropic low stress no damage with fabric 45° to tunnel face.](image)
F.2.5.32 Isotropic Low Stress No Damage with Fabric 0° to the Tunnel Face

The isotropic low stress no damage with fabric 0° to the tunnel face sample is classified as a type 2 chip with 1.2cm chip length and 0.4cm chip depth on the left side and 2.5cm chip length and 0.7cm chip depth on the right side.

Figure F.2.70: FLAC output of deformed grid (left) and plastic shear strain (right) for isotropic low stress no damage with fabric 0° to tunnel face.

F.2.5.33 Isotropic Low Stress No Damage without Fabric

The isotropic low stress no damage without fabric sample is classified as a type 4 chip with 1.3cm chip length and 0.4cm chip depth on the left side and 0.6cm chip length and 1.5cm chip depth on the right side.

Figure F.2.71: FLAC output of deformed grid (left) and plastic shear strain (right) for isotropic low stress no damage without fabric.
F.2.5.34 Isotropic Medium Stress No Damage with Fabric 45° to the Tunnel Face

The isotropic medium stress no damage with fabric 45° to the tunnel face sample is classified as a type 2 chip with 1.5cm chip length and 0.6cm chip depth on the left side and 1cm chip length and 0.4cm chip depth on the right side.

Figure F.2.72: FLAC output of deformed grid (left) and plastic shear strain (right) for isotropic medium no damage with fabric 45° to tunnel face.

F.2.5.35 Isotropic Medium Stress No Damage with Fabric 0° to the Tunnel Face

The isotropic medium stress no damage with fabric 0° to the tunnel face sample is classified as a type 4 chip with 1.5cm chip length and 0.6cm chip depth on the left side and 0.5cm chip length and 0.3cm chip depth on the right side.

Figure F.2.73: FLAC output of deformed grid (left) and plastic shear strain (right) for isotropic medium no damage with fabric 0° to tunnel face.
F.2.5.36 Isotropic Medium Stress No Damage without Fabric

The isotropic medium stress no damage without fabric sample is classified as a type 1 chip with 0.6cm chip length and 1cm chip depth on the left side and 0.5cm chip length and 0.2cm chip depth on the right side.

Figure F.2.74: FLAC output of deformed grid (left) and plastic shear strain (right) for isotropic medium stress no damage without fabric.

F.2.5.37 Isotropic High Stress No Damage with Fabric 45° to the Tunnel Face

The isotropic high stress no damage with fabric 45° to the tunnel face sample is classified as a type 5 chip with 0.4cm chip length and 0.6cm chip depth on the left side and 0.2cm chip length and 0.6cm chip depth on the right side.

Figure F.2.75: FLAC output of deformed grid (left) and plastic shear strain (right) for isotropic high stress no damage with fabric 45° to the tunnel face.
F.2.5.38 Isotropic High Stress No Damage with Fabric 0° to the Tunnel Face

The isotropic high stress no damage with fabric 0° to the tunnel face sample is classified as a type 4 chip with 0.8cm chip length and 0.2cm chip depth on the left side and 1.1cm chip length and 0.5cm chip depth on the right side.

Figure F.2.76: FLAC output of deformed grid (left) and plastic shear strain (right) for isotropic high stress no damage with fabric 0° to the tunnel face.

F.2.5.39 Isotropic High Stress No Damage without Fabric

The isotropic high stress no damage without fabric sample is classified as a type 4 chip with 1.2cm chip length and 0.8cm chip depth on the left side and 0.4cm chip length and 0.2cm chip depth on the right side.

Figure F.2.77: FLAC output of deformed grid (left) and plastic shear strain (right) for isotropic high stress no damage without fabric.
F.2.5.40 Biaxial Very Low Stress No Damage with Fabric 45° to the Tunnel Face

The biaxial very low stress no damage with fabric 45° to the tunnel face sample is classified as a type 2 chip with 0.2cm chip length and 1.6cm chip depth on the left side and 1.3cm chip length and 0.7cm chip depth on the right side.

Figure F.2.78: FLAC output of deformed grid (left) and plastic shear strain (right) for biaxial very low stress no damage with fabric 45° to tunnel face.

F.2.5.41 Biaxial Very Low Stress No Damage without Fabric

The biaxial very low stress no damage without fabric sample is classified as a type 2 chip with 0.6cm chip length and 1.1cm chip depth on the left side and 0.6cm chip length and 1cm chip depth on the right side.

Figure F.2.79: FLAC output of deformed grid (left) and plastic shear strain (right) for biaxial very low stress no damage without fabric.
F.2.5.42 Biaxial Low Stress No Damage with Fabric 45° to the Tunnel Face

The biaxial low stress no damage with fabric 45° to the tunnel face sample is classified as a type 2 chip with 1cm chip length and 0.6cm chip depth on the left side and 1cm chip length and 1.2cm chip depth on the right side.

Figure F.2.80: FLAC output of deformed grid (left) and plastic shear strain (right) for biaxial low stress no damage with fabric 45° to the tunnel face.

F.2.5.43 Biaxial Low Stress No Damage without Fabric

The biaxial low stress no damage without fabric sample is classified as a type 2 chip with 0.9cm chip length and 0.9cm chip depth on the left side and 0.9cm chip length and 0.9cm chip depth on the right side.

Figure F.2.81: FLAC output of deformed grid (left) and plastic shear strain (right) for biaxial low stress no damage with no fabric.
F.2.5.44 Biaxial Medium Stress No Damage with Fabric 45° to the Tunnel Face

The biaxial medium stress no damage with fabric 45° to the tunnel face sample is classified as a type 4 chip with 0.5cm chip length and 0.2cm chip depth on the left side and 1cm chip length and 1.2cm chip depth on the right side.

Figure F.2.82: FLAC output of deformed grid (left) and plastic shear strain (right) for biaxial medium stress no damage with fabric 45° to the tunnel face.

F.2.5.45 Biaxial Medium Stress No Damage without Fabric

The biaxial medium stress no damage without fabric sample is classified as a type 5 chip with 0.5cm chip length and 0.2cm chip depth on the left side and 0.5cm chip length and 0.1cm chip depth on the right side.

Figure F.2.83: FLAC output of deformed grid (left) and plastic shear strain (right) for biaxial medium stress no damage without fabric.
F.2.5.46 Biaxial High Stress No Damage with Fabric 45° to the Tunnel Face

The biaxial high stress no damage with fabric 45° to the tunnel face sample is classified as a type 4 chip with 0.5cm chip length and 0.3cm chip depth on the left side and 0.8cm chip length and 1.2cm chip depth on the right side.

Figure F.2.84: FLAC output of deformed grid (left) and plastic shear strain (right) for biaxial high stress no damage with fabric 45° to the tunnel face.

F.2.5.47 Biaxial High Stress No Damage without Fabric

The biaxial high stress no damage without fabric sample is classified as a type 5 chip with 0.2cm chip length and 0.3cm chip depth on the left side and 0.2cm chip length and 0.2cm chip depth on the right side.

Figure F.2.85: FLAC output of deformed grid (left) and plastic shear strain (right) for biaxial high stress no damage with no fabric.
F.2.6 Two-Cutter Modelling of Southern Aar Granite

The samples of Southern Aar granite for which laboratory and modelling UCS testing were performed were used for two cutter modelling in Subsections F.2.5.1 to F.2.4.7. The estimated stress state described in Section 6.3.2.1 was used to simulate the in-situ stress state (48MPa horizontal, 28MPa vertical and 41MPa out of plane). Subsections F.2.5.8 to F.2.4.11 have two different geological models representing real samples from the Altkristallin modelled under two different stress conditions: the stress condition under which the Altkristallin was excavated (38MPa horizontal, 33MPa vertical and 35MPa out of plane) and the stress condition encountered 500m previous (36MPa horizontal, 32MPa vertical and 35MPa out of plane), estimated in Section 6.3.2.1.

F.2.6.1 Sample GA_a065

The GA_a065 sample is classified as a type 5 chip with 0.8cm chip length and 0.3cm chip depth on the left side and 0.5cm chip length and 0.3cm chip depth on the left side.

Figure F.2.86: FLAC output of deformed grid (left) and plastic shear strain (right) for GA_a065.
F.2.6.2  Sample GA_a169

The GA_a169 sample is classified as a type 2 chip with 1.2cm chip length and 1cm chip depth on the left side and 1.5cm chip length and 0.3cm chip depth on the right side.

Figure F.2.87: FLAC output of deformed grid (left) and plastic shear strain (right) for GA_a169.

F.2.6.3  Sample GA_b011

The GA_b011 sample is classified as a type 4 chip with 0.5cm chip length and 0.3cm chip depth on the left side and 1cm chip length and 0.6cm chip depth on the right side.

Figure F.2.88: FLAC output of deformed grid (left) and plastic shear strain (right) for GA_b011.
F.2.6.4 Sample GA_b023

The GA_b023 sample is classified as a type 4 chip with 1.5cm chip length and 0.3cm chip depth on the left side.

Figure F.2.89: FLAC output of deformed grid (left) and plastic shear strain (right) for GA_b023.

F.2.6.5 Sample GA_b039

The GA_b039 sample is classified as a type 1 chip with 2cm chip length and 0.3cm chip depth on the left side and 1.2cm chip length and 0.3cm chip depth on the right side.

Figure F.2.90: FLAC output of deformed grid (left) and plastic shear strain (right) for GA_b039.
F.2.6.6 Sample GA_b088

The GA_b088 sample is classified as a type 1 chip with 1.5cm chip length and 0.4cm chip depth on the left side and 1cm chip length and 0.5cm chip depth on the right side.

Figure F.2.91: FLAC output of deformed grid (left) and plastic shear strain (right) for GA_b088.

F.2.6.7 Sample GA_b124

The GA_b124 sample is classified as a type 1 chip with 1.5cm chip length and 0.5cm chip depth on the left side and 2cm chip length and 0.8cm chip depth on the right side.

Figure F.2.92: FLAC output of deformed grid (left) and plastic shear strain (right) for GA_b124.
F.2.6.8  Altkristallin Fabric GA_099 Stress State 1

The GA_099 stress state 1 sample is classified as a type 4 chip with 0.4cm chip length and 0.4cm chip depth on the left side and 2.1cm chip length and 2.2cm chip depth on the right side.

Figure F.2.93: FLAC output of deformed grid (left) and plastic shear strain (right) for GA_099 stress state 1.

F.2.6.9  Altkristallin Fabric GA_099 Stress State 2

The GA_099 stress state 2 sample is classified as a type 4 chip with 1.1cm chip length and 0.4cm chip depth on the left side and 1.6cm chip length and 0.8cm chip depth on the right side.

Figure F.2.94: FLAC output of deformed grid (left) and plastic shear strain (right) for GA_099 stress state 2.
F.2.6.10 Altkristallin Fabric GA_013 Stress State 1

The GA_013 stress state 1 sample is classified as a type 4 chip with 1.2cm chip length and 0.4cm chip depth on the left side and 1cm chip length and 0.4cm chip depth on the right side.

Figure F.2.95: FLAC output of deformed grid (left) and plastic shear strain (right) for GA_013 stress state 1.

F.2.6.11 Altkristallin Fabric GA_013 Stress State 2

The GA_013 stress state 1 sample is classified as a type 4 chip with 1cm chip length and 0.2cm chip depth on the left side and 0.5cm chip length and 0.4cm chip depth on the right side.

Figure F.2.96: FLAC output of deformed grid (left) and plastic shear strain (right) for GA_013 stress state 2.
F.2.7 Summary of Goodness of Chip

The following tables contain a summary of the chip type category, chip length and depth under each cutter and high, medium and low chip rating based on chip area. The categories are as follows:
Chip area > 3cm² = high (H) with very high (very H) for samples with Chip area >7cm²
1.5cm² > Chip area > 3cm² = medium (M)
Chip area < 1.5cm² = low (L)

F.2.7.1 Parametric Samples without Stress

Table F.2.2: Goodness of chip summary for parametric samples without stress.

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### F.2.7.2 Samples with Stress

Table F.2.3: Goodness of chip summary for parametric samples with stress.

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### F.2.7.3 Southern Aar Granite and Altkristallin Samples

Table F.2.4: Goodness of chip summary for Southern Aar granite samples.

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<th>Sample</th>
<th>Chip Type</th>
<th>Left Cutter Fracture</th>
<th>Right Cutter Fracture</th>
<th>Rating</th>
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<td>Depth</td>
<td>Length</td>
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Appendix F.3 : Geomechanical Characterisation of Southern Aar Granite Between Tunnel Metre 116428 and 116812

F.3.1 Introduction

The geological record from Appendix B.4 was used as the basis for this Appendix. The F Factors were quantified by verification and calibration in Chapters 3, 5 and 6. The spalling sensitivity, and Chipping Resistance were calculated from the F Factors and the values were digitised. This Figure is found on the CD accompanying the thesis.